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GROUP OF MILLS AND TAILING PILES, WEBB CITY DISTRICT.

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## BULLETINS WANTED

The Institute has no available copies of *Bulletin* No. 7, January, 1906, and *Bulletin* No. 105, September, 1915. The sum of \$0.50 per copy will be paid for these *Bulletins*.

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## ST. LOUIS MEETING

Oct. 8-13, 1917

Zinc is just now a very important metal. Its use in cartridges and shrapnel brass are in the public eye. But its uses after the war are no less important. Brass as an engineering material for electrical machinery, journal bearings, and plumbing fixtures takes fully 50 per cent. of the production, and its use in these lines would have been increased in the last few years but for the price. Its use for barbed wire is probably more in war time than in peace. In the composition of paints, it is becoming more and more necessary. One of the forms of pigment, lithopone, is the base of nearly all interior wall paints; of all the sanitary enamels for bath-rooms and bath-tubs, and is now in great demand. Zinc oxide is also increasing in use for paint, and for manufacture of linoleum.

When we realize that this district—Missouri, Oklahoma, Kansas and Arkansas—produces now over one-third of the zinc ore of the world, it is at once seen that a meeting where the geology, mining, concentration

and smelting of this ore will be considered, and visits made to Joplin, Missouri and Miami, Okla., as well as to some of the modern smelters where the sulphurous gases are turned into sulphuric acid, will not only be interesting, but at this time important. We want to show you the immense low-grade mines of Joplin and Webb City, Mo. Some mines produce ores with only 1 per cent. of zinc. Likewise, we want to show you the high-grade mines of Miami, Okla., where some of the mines are producing ore that nets 20 per cent. zinc.



DAYLIGHT MINING IN JOPLIN DISTRICT.

The problems connected with the concentration are not so simple as they once looked. It is easy to get out the first 80 per cent. of the value, but the treatment of the slimes is as yet a problem. Much is hoped for in the comparatively new process of flotation. The precise place of the new electrolytic process of recovering zinc is not yet known. It has proven successful in Montana, and may be equally valuable in this district.

We invite to our meeting next October all who can give intelligent answer to the question "How to get the last few per cent. of zinc out of the ore?"

J. D. ROBERTSON,  
*Chairman, Publicity Committee.*



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| American Institute of Electrical Engineers: |   | IRA N. HOLLIS, <i>President</i> ,         |
|   |   | CALVIN W. RICE, <i>Secretary</i> .        |
|   |   | H. W. BUCK, <i>President</i> ,            |
|   |   | C. A. ADAMS, <i>Chairman of Standards</i> |
|   |   | <i>Committee</i> .                        |

It is intended that this sub-committee shall be the means of putting the societies and the individual members thereof into relations with the Council of National Defense so that the services offered by the societies may be available to the Council.

The Council of National Defense advises with the Army and Navy Departments and other departments of the National Service.

It receives advice regarding inventions from the Naval Consulting Board. It receives advice regarding Research from the National Research Council.

Through its Committee on Science and Research, the Council of National Defense calls on the individual members and resources of the National Engineering Societies.

Some of the sub-committees and boards appointed by the Council and Advisory Commission are:

*Munitions Standards Board*, under the general direction of the Munitions and Manufacturing Committee, dealing, in coöperation with the Government, with drawings and specifications, jigs, fixtures, etc., Frank A. Scott, Chairman. An important sub-committee of this board, of which J. E. Otterson is chairman, relates to small arms and ammunition.

*General Munitions Board*, under the direction of Frank A. Scott. The secretary is Chester C. Bolton. The purpose of this board is to assume the prompt equipment and arming of whatever forces may be called into the service of the country. This board, which comprises in its membership seven representatives of the Army and eight of the Navy, is expected to undertake the work of coördinating the purchases for the Army and Navy, but it is not intended that it shall have authority to issue purchase orders or to bind the Government in contracts for purchase.

*Commercial Economy Board*, A. W. Shaw, Chairman, designed to meet in advance problems of war-time distribution and to bring business men together in a scheme of voluntary coöperation to eliminate waste by adopting commercially efficient methods.

*Food Board*, Herbert C. Hoover, Chairman. It is proposed that this board shall take such guiding action as may be necessary to increase the production of food, to conserve as well as to increase farm labor, to instill a willingness for the elimination of waste, to control the exportation of food and to prevent speculation or undue profits.

### THE NAVAL CONSULTING BOARD

\* THOMAS ALVA EDISON, *President*,

PETER COOPER HEWITT, *Vice-President*,

\* WILLIAM L. SAUNDERS, *Chairman*,

\* THOMAS ROBINS, *Secretary*.

\* LAWRENCE ADDICKS,  
BION J. ARNOLD,  
L. H. BAEKELAND,  
HOWARD E. COFFIN,  
ALFRED CRAVEN,  
WILLIAM LEROY EMMET,  
ANDREW MURRAY HUNT,  
M. R. HUTCHISON,  
B. G. LAMME,  
HUDSON MAXIM,

SPENCER MILLER,  
\* JOSEPH W. RICHARDS,  
ANDREW L. RIKER,  
MATTHEW BACON SELLERS,  
ELMER A. SPERRY,  
FRANK J. SPRAGUE,  
\* BENJAMIN B. THAYER,  
ARTHUR GORDON WEBSTER,  
\* W. R. WHITNEY,  
ROBERT S. WOODWARD.

This body passes upon all inventions for the Army, Navy, or other Departments of the National Government, and develops them in the new Government Laboratory installed for that purpose. It reports to the Council of National Defense.

The Associate Boards of the Naval Consulting Board, organized in each State and Territory will remain in existence for work now being arranged.

### THE NATIONAL RESEARCH COUNCIL

This body was appointed recently by the President of the National Academy of Sciences, at the request of President Wilson. The National Academy of Sciences was created by Congress during the Civil War to assist the Government.

The organization and operating expenses, but not the research expenses, of the National Research Council are paid by the Engineering Foundation of the United Engineering Society. The National Research Council is composed as follows:

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\* A. I. M. E. member.

GEORGE E. HALE, *Chairman*,  
 \* CHARLES D. WALCOTT, *First Vice-Chairman*,  
 GANO DUNN, *Second Vice-Chairman*,  
 CARY T. HUTCHINSON, *Secretary*.

CARL ALSBERG,  
 JOSEPH AMES,  
 LEO H. BAEKELAND,  
 MARSTON TAYLOR BOGERT,  
 JOHN A. BRASHEAR,  
 WALTER B. CANNON,  
 JOHN J. CARTY,  
 RUSSELL H. CHITTENDEN,  
 JOHN M. CLARKE,  
 HOWARD E. COFFIN,  
 EDWIN G. CONKLIN,  
 JOHN M. COULTER,  
 WILLIAM CROZIER,  
 WILLIAM M. DAVIS,  
 HERBERT H. DONALDSON,  
 RALPH EARLE,  
 SIMON FLEKNER,  
 JOHN R. FREEMAN,  
 JAMES D. GATEWOOD,  
 HOLLIS GODFREY,  
 WILLIAM C. GORGAS,  
 W. F. M. GOSS,  
 ROBERT S. GRIFFIN,  
 CLEMENS HERSCHEL,  
 WILLIAM H. HOLMES,  
 \* HERBERT CLARK HOOVER,  
 W. W. KEEN,

\* VAN H. MANNING,  
 FRANKLIN H. MARTIN,  
 CHARLES F. MARVIN,  
 JOHN C. MERRIAM,  
 A. A. MICHELSON,  
 ROBERT A. MILLIKAN,  
 E. H. MOORE,  
 ARTHUR A. NOYES,  
 RAYMOND PEARL,  
 \* E. C. PICKERING,  
 MICHAEL I. PUPIN,  
 \* CHARLES F. RAND,  
 THEODORE W. RICHARDS,  
 C. E. SKINNER,  
 \* GEORGE OTIS SMITH,  
 GEORGE O. SQUIER,  
 LEWIS B. STILLWELL,  
 S. W. STRATTON,  
 AMBROSE SWASEY,  
 DAVID W. TAYLOR,  
 ELIHU THOMSON,  
 \* C. R. VAN HISE,  
 VICTOR C. VAUGHAN,  
 ARTHUR G. WEBSTER,  
 WILLIAM H. WELCH,  
 \* W. R. WHITNEY,  
 ROBERT W. WOOD,  
 ROBERT M. YERKES.

The National Research Council has constituted Research Committees on: Physical Sciences and Chemistry, Mathematics, Medicine, Hygiene, Agriculture, Engineering, etc. Gano Dunn is Chairman of its Committee on Engineering. The other members are:

GEORGE F. SWAIN  
 EDGAR C. MARBURG  
 CLEMENS HERSCHEL

} Representing the American Society of Civil Engineers.

CHARLES F. RAND  
 GEORGE K. BURGESS  
 POPE YEATMAN  
 ALBERT SAUVEUR

} Representing the American Institute of Mining Engineers.

JOHN A. BRASHEAR  
 W. F. M. GOSS  
 JOHN R. FREEMAN  
 HOLLIS GODFREY  
 HOWARD E. COFFIN  
 AMBROSE SWASEY  
 W. F. DURAND  
 CHAS. D. YOUNG

} Representing the American Society of Mechanical Engineers.

JOHN J. CARTY  
 FRANK B. JEWETT  
 CLAYTON H. SHARP  
 GANO DUNN  
 C. E. SKINNER  
 MICHAEL I. PUPIN  
 S. W. STRATTON  
 ELIHU THOMSON

} Representing the American Institute of Electrical Engineers.

LEWIS B. STILLWELL

} Representing the American Institute of Consulting Engineers.

The National Research Council reports to the Council of National Defense on research matters. It coöperates with the Naval Consulting Board.

### OFFICERS' RESERVE CORPS

An Engineer Officers' Reserve Corps was suggested by General Leonard Wood at a luncheon tendered him by the Secretaries of the A. S. C. E., the A. I. M. E., the A. S. M. E., and the A. I. E. E. The project was worked out and promoted by a Joint Conference Committee consisting of:

WILLIAM BARCLAY PARSONS, *Chairman*, Representing the A. S. C. E.

ARTHUR S. DWIGHT, Representing the A. I. M. E.

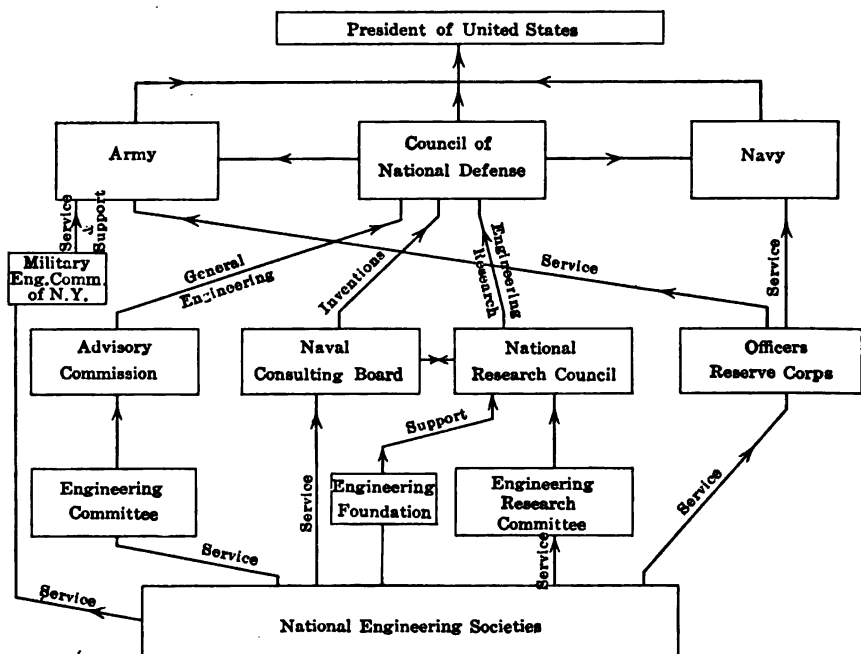
WILLIAM H. WILEY, Representing the A. S. M. E.

BION J. ARNOLD, Representing the A. I. E. E.

RALPH D. MERSHON, Representing the A. I. C. E.

The project was expanded by Congress to include an Officer's Reserve Corps for all departments of the Army and Navy, including the Corps of Engineers, and became law on July 1, 1916.

Application blanks for Commissions in the Engineer Officers' Reserve Corps may be obtained from the Secretaries of the Societies mentioned above, or from the War Department, Washington, D. C.



**MEMBERS OF THE INSTITUTE  
IN THE  
ENGINEER OFFICERS' RESERVE CORPS\***

**MAJORS**

CHAPMAN, R. H., Washington, D. C.,  
DWIGHT, ARTHUR S., New York, N. Y.,  
HURD, RUKARD, St. Paul, Minn.,  
MUDD, SEELEY W., Los Angeles, Cal.,  
WEINBERG, GEORGE S., New York, N. Y.

**CAPTAINS**

GRUNSKY, CARL E., JR., San Francisco, Cal.  
HEGDEM, ALFRED G., Tulsa, Okla.,  
IRVING, J. D., New Haven, Conn.,  
KNAPP, ARTHUR, Ardmore, Pa.,  
LACROIX, M. F., Lynn, Mass.,  
PARKS, HENRY M., Corvallis, Ore.,  
SACKET, CHARLES T., Livingston, Mont.,  
STEHLI, H. J., New York, N. Y.

**FIRST LIEUTENANTS**

BARLING, H. B., Glendon, N. C.,  
DUNN, THEODORE S., Dubois, Pa.,  
GOODSPEED, G. E., JR., Corvallis, Ore.,  
GRUNOW, W. R., Morenci, Ariz.,  
HAGUE, WILLIAM, Grass Valley, Cal.,  
HINES, P. R., El Paso, Tex.

**SECOND LIEUTENANT**

ROUDABUSH, M. A., Osterburg, Pa.

**Resolution Passed by Military Engineering Committee of New York**

The following resolution was passed by the Military Engineering Committee of New York on April 12, 1917. Copies are being sent to each member of Congress and to each of the five members in every State forming the committee representing the five National engineering societies which carried on the industrial census under the direction of Howard Coffin. The Committee is urging every man to wire his Congressman and his Senator, asking him to do everything in his power to obtain favorable action by Congress on this resolution:

Whereas in any large army to be created for the war, engineer troops will be needed in large numbers, and

Whereas they should be officered by skilled engineers only, and

Whereas the composition and training of such troops differ so much from the composition and training of non-technical troops that they should be organized under one central authority rather than under many scattered authorities.

Now, therefore, be it resolved:

That the members of the Senate and House of Representatives be requested to insist that all new units of engineer troops be organized and maintained by the War Department and not by the individual States.

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\* According to the records of April 10. This list is no doubt incomplete, and we would appreciate information concerning members who are engaged in any military service.

## PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members)

The following is a partial list of members and guests who called at Institute headquarters during the period Mar. 10, 1917, to Apr. 10, 1917.

|  |   |
|--|---|
| George S. Blair, Pittsburgh, Pa.       | E. P. Mathewson, Toronto, Canada.         |
| H. M. Boylston, Cambridge, Mass.       | T. Poole Maynard, Atlanta, Ga.            |
| F. J. Brule, Toronto, Canada.          | W. J. Mead, Madison, Wis.                 |
| M. B. Cohen, Miami, Ariz.              | D. G. Miller, Denver, Colo.               |
| W. M. Corse, Buffalo, N. Y.            | P. N. Moore, St. Louis, Mo.               |
| F. G. Cottrell, Washington, D. C.      | R. H. Morris, Ansted, W. Va.              |
| E. C. Cross, Cleveland, O.             | W. J. Morrison, Toronto, Canada.          |
| Nelson Dickerman, San Francisco, Cal.  | M. F. Ortin, Petrograd, Russia.           |
| F. N. Flynn, Chuquicamata, Chile.      | G. W. Paymal, New York, N. Y.             |
| F. L. Garrison, Philadelphia, Pa.      | Edward H. Perry, Boston, Mass.            |
| J. S. Grasty, Charlottesville, Va.     | O. G. Petersen, Somerset, Ky.             |
| H. D. Hibbard, Plainfield, N. J.       | H. C. Price, Bartlesville, Okla.          |
| H. L. Hollis, Chicago, Ill.            | C. A. Richardson, Toronto, Canada.        |
| F. H. Jackson, Marysville, Utah.       | Hallet R. Robbins, Vancouver, B. C., Can. |
| W. Strickler Jones, Silver City, N. M. | A. H. Rogers, Boston, Mass.               |
| E. B. Kirby, St. Louis, Mo.            | H. G. Smith, Princepulca, Nic.            |
| J. M. Lovejoy, Electra, Tex.           | Edwin G. Steele, Van Horn, Tex.           |
| A. F. Lucas, Washington, D. C.         | S. A. Taylor, Pittsburgh, Pa.             |
| Edwin Ludlow, Lansford, Pa.            | A. P. Thompson, Salt Lake City, Utah.     |
| E. Francis McCrossin, Birmingham, Ala. |   |

Milton A. Allen is now engaged in the New York office of Messrs. Arthur L. Pearse & Co.

Percy E. Barbour, managing editor of the *Engineering & Mining Journal*, has been appointed, by Governor Whitman, Deputy Superintendent of Police of the State of New York.

F. K. Brunton, who was metallurgist and assistant superintendent of the British Columbia Copper Co. until the beginning of the war, and who has recently been assistant superintendent of the Arizona Smelting Co., has been promoted to the position of superintendent with the latter company.

C. E. Bunker has been appointed superintendent of the Bunker Hill mine, Amador City, Cal.

Edward B. Durham is engineer for the Standard Chemical Co., Canonsburg, Pa. The company mines carnotite ore in Colorado and produces vanadium and uranium as ferro-alloys and salts. Mr. Durham is engaged in the designing and installing of improvements at the extraction plant.

George A. Easley has resigned as manager of The Olla de Oro Gold Min. Co., in order to devote all his time to his tungsten interests in Bolivia. Until Aug. 1, his address will be Lees Summit, Mo., after which he will be at his office in La Paz, Bolivia.

F. N. Flynn, general superintendent of the Chile Exploration Co. at Chuquicamata, Chile, has severed his connection with that company.

Alfred K. Friedrich has been transferred to the mine of the Buckeye Coal Co., Carmichaels, Greene Co., Pa.

**Walter E. Gaby** has been appointed geologist for the Primos Chemical Co., Vanadium, Colo., and is looking after the development of its property.

**Herman Garlichs** has resigned his position with the St. Louis Smelting & Refining Co., and will open an office in New York in the fall.

**Justice Grugan** has been appointed consulting engineer to the Grasselli Chemical Co. and is directing some of their diamond-drill explorations.

**Edmund Juessen** has been appointed general manager for the New Almaden Company, at New Almaden, California.

**George Kingdon** has been made superintendent at the United Verde Extension mines, Ariz.

**W. H. Landers** has resigned as manager for the New Almaden Co. and will offer his services to the Government.

**Frank C. Laurie** has been appointed general superintendent for the Freeport and Tampico Fuel-Oil Corporation, with headquarters at Tampico, Mexico.

**William W. Logan** has accepted a position with the Willow Creek Mining Co., Knik, Alaska, and will be there until about Nov. 1, 1917.

**Walter G. Perkins** has opened an office at 462 Mills building, San Francisco.

**W. A. Pomeroy** and **W. R. Hamilton** have formed a partnership with offices in the Hobart building, San Francisco.

**Jesse C. Porter** and **John B. Stewart** are now in charge of the business of the C. L. Constant company of New York, in the National Bank building at Calle Cuba, Havana, Cuba.

**H. J. Rahilly** has resigned as mining engineer for the Bureau of Mines to accept a position with the Anaconda Copper Mining Co.

**Rastus S. Ransom** sailed on Apr. 11, for Vigo, Spain, on his way to Visen, Portugal, to institute construction work for the Dias Mining Corporation.

**C. R. Richards**, Professor of Mechanical Engineering at the University of Illinois, and head of the department since 1911, has been appointed Dean of the College of Engineering and Director of the Engineering Experiment Station to succeed Dr. W. F. M. Goss who has resigned to become President of the Railway Car Manufacturer's Association of New York.

**John A. Root**, of the technical staff of the Anaconda Copper Mining Co., Washoe Reduction department, has been promoted to the position of chief chemist.

**W. L. Saunders** has been elected a member of the board of the New York and Honduras Rosario Mining Co.

**Francis S. Schimerka**, who has been in charge of the leaching operations at the Shannon Copper Co., Clifton, Ariz., for the last four years, has resigned his position.

**J. G. Scrugham** has been appointed State Engineer of Nevada to succeed William Kearney, who has held the position for the last five years. Mr. Scrugham was dean of the mechanical school at the State University at the time of his appointment.

**H. J. Sheafe** has been appointed superintendent of the Globe mine, in Trinity county, California.

**J. A. Singmaster**, formerly general superintendent of the Palmerton, Pa., plant of the New Jersey Zinc Co., has been transferred to the New York office to take charge of metallurgical investigations.

**Francis P. Sinn** has been made general superintendent of the Palmer-ton, Pa., plant of the New Jersey Zinc Co.

**D. R. Thomas** has resigned as manager for the Davidson Gold Mines, Ltd., at Porcupine, Ont., and is temporarily in New York.

**Henry N. Thomson** has resigned as metallurgist of the United Verde Copper Co., Clarkdale, Ariz., and is engaged in consulting practice at 918 South Kingsley Drive, Los Angeles, Cal.

**Andrew M. Tweedy** is resident manager of the Zaruma mines in Ecuador.

**A. J. Underwood** is now manager of the Lluvia de Oro mines, Chihuahua, Mex.

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### ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members)

Graduate mining and metallurgical engineer now open for engagement. Member, 38 years old, married, 19 years' experience in United States, Europe, South America, Africa, in exploration work, surveying, assaying, mining, milling, ore-dressing, construction and management. Four languages mastered. No. 355.

Member, aged 35, technical graduate, desires position as superintendent or assistant superintendent of spelter or zinc oxide furnaces, or both. Gas producer experience, zinc metallurgical and laboratory experience. Can handle men. No. 356.

Member, also Assoc. Mem. Am. Soc. of C. E.; excellent technical education, and best of references covering twelve years in responsible charge of work; experienced in surveys, construction, prospecting, mine examination and development. and irrigation work; speak Spanish fluently, and can write a good report. Good executive. Available during June. Age 34; married. No. 357.

Member, aged 31. Five years' experience in zinc smelting, as chief chemist and superintendent; two years foreman and superintendent zinc refinery. Thorough assayer and chemist, both in theory and practical experience. Experienced cement chemist. Married. Reference. At present employed. No. 358.

Executive position wanted by an American graduate mining engineer (36), member A. I. M. E. and A. S. M. E. Fourteen years' experience in copper, gold, silver and coal mining and general engineering work. At present employed. New York interview if desired. No. 359.

Member, graduate mining engineer, nine years' experience in mining and milling. Young, energetic, capable man. Desires connection with mine operator handling mine or prospect under development or lease and bond. Can direct and carry out such work, assuming entire responsibility for operations. Small salary and some interest preferred. Best references as to character, responsibility and ability. Personal interview. No. 360.



**ALASKA MINING AND ENGINEERING SOCIETY**

A special meeting of the Alaska Mining and Engineering Society, held at Perseverance Mine of the Alaska Gastineau Mining Co. on Mar. 17, 1917, was attended by 46 members and about 80 guests, among whom were members of the Alaska Territorial Legislature.

After an interesting trip through the mine in the afternoon, the members and guests assembled for dinner at the Perseverance Dining Room. After the dinner G. T. Jackson, assistant manager of the company, as president of the society, called the meeting to order. After the reading of the minutes, he welcomed the guests and then gave a short history of Perseverance Mine. B. L. Thane, manager of the company, briefly outlined its general organization.

The following papers were read by members of the society and members of the staff of the Perseverance Mine: "The Organization of the Mine Staff and the Methods of Mining," by B. B. Nieding, Mine Supt.; "Tramming Equipment and Costs," by R. R. Van Valkenburgh; "The Surveying of the Mine," by R. L. Healey, Mine Engineer; "The Electrical Equipment of the Mine," by W. S. Pullen, Supt. of Power Supply; "The System of Mine Accounting," by E. J. Doherty, Mine Accountant; and "The Mine Buildings," by D. J. Argall, Asst. Mine Supt. W. T. Tolch, Transportation Supt., described briefly the general transportation of the ore to the mill and the equipment used.

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**LOCAL SECTION NEWS****BOSTON SECTION**

R. L. AGASSIZ, *Chairman*,

J. G. CARLETON, *Vice-Chairman*,

E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Cambridge, Mass.

W. E. C. EUSTIS,

G. A. PACKARD.

The thirty-ninth meeting of the Boston Section was held at the Engineers' Club on Monday evening, Dec. 4, 1916.

Thirty-three members and guests sat down to dinner.

There were present as section guests, Dr. L. D. Ricketts, and Prof. Bradley Stoughton, President and Secretary of the parent society, and each was called upon to speak.

Mr. Stoughton spoke of the growth of the Institute during the past year and described the very numerous and important activities, of which the junction of the American Society of Civil Engineers with the other three societies is perhaps the most important.

Dr. Ricketts, the advertised speaker of the evening, spoke on Recent Progress in Mining and Metallurgy of Copper Ores. In discussing the improvements in mining, he traced the mining methods from square-set mining to the modified Ohio System now so successfully used at Inspiration.

In metallurgy, he described the growth of reverberatory smelting and allied problems such as the utilization of the waste heat for generating steam. He laid stress on the importance of sampling and bedding of

copper ores and on a careful technical study of the details of the metallurgical operations. He told how attention to the above details had reduced the uncertainty due to the unaccounted losses.

He next told of the development of the Inspiration flow sheet and of the great saving in cost due to the adoption of flotation.

An outline of the work being done at Ajo was given and the whole thing characterized as a \$5,000,000 experiment. He ended with a plea for greater freedom in exchange of ideas between engineers.

The fortieth meeting was held at the Engineers' Club on Tuesday evening, Jan. 2, 1917.

After the usual dinner, the Secretary was authorized to invite the members of the Engineers' Club to attend the next meeting of the section to listen to an illustrated talk describing the Operation of the Calumet & Hecla Mine and Mills.

Professor Waldemar Lindgren, the speaker of the evening, then gave an informal talk on the deposition of the various forms of silica. He classified these as opal, chalcedony and quartz. Opal, which results from the hardening of colloidal silica, is deposited near the surface from hot springs. It is found rarely to inclose sulphides or gold, but may be associated with cinnabar deposits. On account of its water content, it is a very unstable form of silica and can not exist at high temperatures. Chalcedony, on the contrary, although comparatively unimportant, is very stable. It does occasionally enclose gold and may fill cavities or replace other minerals. It contains very little or no water, and crystallizes in fibrous form. Quartz is thought to be deposited several hundred to one or two thousand feet below the surface and to differ in character according to the depth at which it is deposited. The fine-grained varieties are considered to have been deposited near the surface from either silica jelly or from dilute solutions of silica. Such quartz often shows a banded structure due to its deposition in layers. Specimens from El Oro and Republic, Wash., were shown as illustrating this variety. The coarse-grained massive varieties are considered to be deposited at considerable depths. Specimens from Grass Valley, Cal., and Nova Scotia illustrated this type. Attention was called to the great similarity between deep-seated quartz veins and pegmatite dykes. Such quartz often encloses sulphides and gold. Lenticular veins are common but the reason for this shape is not altogether certain.

After a discussion participated in by Messrs. Lane, Graton, Warren and Stantial, the meeting adjourned.

Eighteen members and guests were present.

The forty-first meeting was held at the Engineers' Club on Monday evening, Feb. 5, 1917.

After the dinner, the 24 members and guests who were present adjourned to the large assembly hall where they were joined by members of the Engineers' Club who had been invited to attend the meeting.

James MacNaughton, General Manager of the Calumet and Hecla Mining Co., gave an interesting description of the operation of the Calumet and Hecla mine and mills. Many excellent slides were shown, illustrating the various phases of the operations.

After numerous questions had been answered by Mr. MacNaughton, the meeting adjourned.

The forty-second meeting was held at the Engineers' Club on Monday evening, Mar. 5, 1917. Eighteen members and guests were present.

After the dinner the Treasurer submitted his report, which was ordered accepted and placed on file.

The officers named above were elected for the ensuing year.

It was moved that the Secretary and Chairman send a telegram to the President assuring him of the support of the section at this time. After discussion, the members present voted affirmatively on the following motion: That the Executive Committee be authorized and instructed to send to the President of the United States a message pledging him the sympathy and support of this section of the A. I. M. E. in such course as may be needful to protect the rights and the honor of the American people.

S. W. Wilder, President of the Merrimac Chemical Co., the guest of the evening, was next introduced by the Chairman, Mr. Eustis. Mr. Wilder spoke of the sources and uses of alumina, which includes not only the manufacture of the metal aluminum, but also the alums used in the sizing of paper and purification of water, of aluminum chloride used in carbonizing of wool, also of C. T. S., i.e., cream of tartar substitute. After considerable discussion, the meeting adjourned.

The forty-third meeting was held at the Engineers' Club on Monday evening, Apr. 2, 1917.

In the absence of Mr. Agassiz, Mr. Carleton presided. After the dinner the Secretary read a telegram which was sent to the President of the United States in accordance with the vote of the last meeting. The message was as follows:

Boston, Mass.,  
March 6, 1917.

To the President,  
Executive Mansion,  
Washington, D. C.

The members of the Boston Section of the American Institute of Mining Engineers at its annual meeting last night, instructed me by an unanimous vote, to communicate to you their sympathy and loyal support in any course you may consider needful to protect the rights and honor of the American people.

Signed, W. E. C. EUSTIS, *Chairman*.

Charles Paige Perin, Consulting Engineer for the Tata Iron and Steel Co., the guest of the evening, was then called upon. He told of the growth of India's iron and steel industry from his first connection with it. The life and customs of the people, and the company's welfare work were also included, and the talk was illustrated with a very good selection of lantern slides. The fact that they are able to produce pig-iron at a cost of but a trifle more than \$5 a ton is significant.

Mr. Perin answered questions after the close of his address until the meeting adjourned at 10:00 p.m.

E. E. BUGBEE, *Secretary*.

## ARIZONA SECTION

Copy of a letter sent to the members of the Arizona Section:

Owing to the conditions which the country is facing, and their effect on the Southwest in particular, your Executive Committee has decided to postpone indefinitely the meeting of the Section scheduled for April 27 and 28, 1917, at Ajo, Arizona.

As there have been no further nominations for the officers of the Section for the coming year, your retiring Chairman, Mr. Gerald Sherman, has authorized me to cast a ballot for the ticket nominated by the Nominating Committee, as follows:

For Chairman, P. G. Beckett, Globe, Arizona.

For First Vice-Chairman, Norman Carmichael, Clifton, Arizona.

For Second Vice-Chairman, J. C. Greenway, Bisbee, Arizona.

For Executive Committee, F. W. MacLennan, Miami, Arizona; L. O. Howard, Miami, Arizona; R. E. Tally, Jerome, Arizona; L. S. Cates, Ray, Arizona.

The next meeting of the Section will be called at the pleasure of the Chairman-elect, P. G. Beckett.

ARTHUR NOTMAN, *Secretary-Treasurer*.

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### PUGET SOUND SECTION

SIMON H. ASH, *Chairman*,

I. F. LAUCKS, *Vice-Chairman*,

CHARLES SIMENSTAD, *Secretary-Treasurer*, 425 Lyon Bldg., Seattle, Wash.

GLENVILLE A. COLLINS,

JOHN N. POTT.

The Puget Sound Section met at the Arctic Club, Seattle, Wash., on Apr. 14, 1917.

Seventeen members and eleven guests were present.

While dinner was being served, the following responded with short toasts: H. H. Sanderson, Geo. W. Ladley, C. W. Willette, Walter A. Horne, John A. Davis, L. A. Levensaler, G. W. Evans, F. C. Greene and Percy E. Wright.

Dinner having been served, the meeting was called to order.

Reading of minutes of previous meeting were suspended so that as much time as possible could be allowed for lectures and discussions.

A committee of three was appointed by the chairman to take full charge of the entertainment of the President and Secretary of the Institute during their visit in Seattle.

Following conclusion of business meeting, John A. Davis, E. M., Superintendent of Bureau of Mines Experimental Station at Fairbanks, Alaska, read a paper on The Proposed Work of the Station, illustrated with slides.

H. H. Sanderson, Mine Safety Engineer, lectured on Coal Mine Fires.

F. R. VanCampen lectured on Mining and Milling Practice at the Beatson Mine, LaTouche, Alaska.

After discussion, the meeting adjourned.

CHAS. SIMENSTAD, *Secretary*.

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### AFFILIATED STUDENT SOCIETY NOTES

The Mining and Geological Society of Lehigh University held its annual election of officers on Apr. 12, 1917, followed by a very pleasant and profitable meeting and discussion. The officers elected for the next collegiate year are as follows:

President, Leonard Sargeant,  
Vice-President, Wm. E. Tizard,  
Secretary, Joseph A. Holmes, 2d,  
Treasurer, C. G. Gilman,  
Curators, Shu Choe, Rudolph Repko.

President McCann, 1917, then made a short address, and introduced Dr. Charles P. Berkey, Professor of Geology at Columbia University, the speaker of the evening. Dr. Berkey gave a very interesting and instructive talk on the Catskill Aqueduct and the problems that were encountered in its construction, laying special emphasis on the phases where the geology of the region was of primary importance to the engineers. The talk was illustrated by a large number of excellent slides, and was greatly enjoyed by the members of the society and their guests, the Civil Engineers.

JOSEPH A. HOLMES, 2d, *Secretary*.

The School of Mines Society, University of Minnesota, on March 15 had the pleasure of hearing a talk by Mr. E. C. Harder, on Recent Mining Developments in South America. Mr. Harder spoke from first-hand knowledge of the subject.

On April 19, the following officers were elected for the year 1917-18:

President, H. K. Armstrong,  
Vice-President, R. W. Allard,  
Secretary-Treasurer, J. O. Hosted,  
Editor of the Bulletin, James Stark,  
Assistant Editor, Henry Chadbourne.

H. K. ARMSTRONG, *Sec.-Treas.*

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### California's Mineral Resources for War Purposes

The State Mining Bureau of California under the direction of Fletcher Hamilton, State Mineralogist, is starting a field campaign to report on the economic minerals of California, which have an important industrial and military bearing on the present war situation. Particular interest and value attaches at the present moment to our available supplies of chrome, coal, iron, magnesite, manganese, molybdenum, quicksilver and tungsten.

Trained geologists and engineers of the Bureau's staff are being sent out to obtain information with relation to the latest developments.

Advance notices will be sent to the local newspapers, from time to time, of the arrival of the engineers in the various districts.

All persons interested in the minerals mentioned are earnestly urged to coöperate with the State Mining Bureau by meeting the field men while in their districts or by communicating direct with the State Mineralogist at his office in the Ferry Building, San Francisco.

The State Mining Bureau has just issued *Bulletin* No. 72 on The Geologic Formations of California. It was written by Prof. James Perin Smith of Stanford University, and contains a complete bibliography and resumé of all of our present knowledge relative to the geological formations occurring in California. It also contains much previously unpublished data, and six tables showing geologic columns. It covers 41 pages.

This bulletin is published in conjunction with the Geologic Map of California, recently issued, and is sold separately—price 25¢, postage prepaid.

Address State Mining Bureau, Ferry Bldg., San Francisco, or State Mining Bureau, 520 Union League Bldg., Los Angeles.

### FORTHCOMING MEETINGS OF SOCIETIES

| Organisation  | Place                | Date<br>1917 |
|---|----------------------|--------------|
| Electric Power Club.....                                | Hot Springs, Va.     | June 11-14   |
| Society for the Promotion of Engineering Education..... | Evansville, Ind.     | June 19-22   |
| American Institute of Chemical Engineers.....           | Buffalo, N. Y.       | June 20-22   |
| American Institute of Electrical Engineers.....         | Hot Springs, Va.     | June 25      |
| American Society for Testing Materials.....             | Atlantic City, N. J. | June 26-30   |
| Mine Inspectors Institute of U. S. A. ....              | Indianapolis, Ind.   | July 9-11    |
| National Association of Stationary Engineers...         | Evansville, Ind.     | Sept. 10-15  |
| National Exposition of Safety and Sanitation..          | New York City        | Sept. 10-15  |
| American Iron, Steel & Electrical Engineers...          | Philadelphia, Pa.    | Sept. 10-14  |
| American Institute of Metals.....                       | Boston, Mass.        | Sept. 24-29  |
| American Foundrymen's Association.....                  | Boston, Mass.        | Sept. 24-29  |
| Chemical Industries, Third National Exhibition.         | New York City        | Sept. 24-29  |
| American Electro-Chemical Society.....                  | Pittsburgh, Pa.      | Oct. 3-6     |
| American Institute of Mining Engineers.....             | St. Louis, Mo.       | Oct. 8-13    |
| American Gas Institute.....                             | Washington, D. C.    | Oct. 16-19   |
| American Institute of Architects.....                   | Philadelphia, Pa.    | Dec. 26-29   |
| American Society of Mechanical Engineers.....           | New York City        | Dec. 26-29   |
| American Mining Congress State Chapter.....             | Phoenix, Ariz.       | Dec. 26-29   |

**LIBRARY**

AMERICAN SOCIETY OF CIVIL ENGINEERS

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

HARRISON W. CRAVER, Director

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

**LIBRARY ACCESSIONS****PARTIAL LIST CLASSIFIED BY SUBJECTS****Mining, Metallurgy and Chemistry**

- COAL TAR AND AMMONIA. Ed. 5, 3 vols. By George Lunge. New York, 1916.  
ELEMENTS OF WATER GAS. By J. Stephenson. Columbia, S. C., 1916.  
HIGHEST DEVELOPMENT OF THE CYANIDING ART. Detroit, n. d. (Gift of Koering Cyaniding Process Company).  
MINING MANUAL AND MINING YEAR BOOK, 1917. By Walter R. Skinner. London, 1917.  
OZONE: ITS MANUFACTURE, PROPERTIES AND USES. By A. Vosmaer. New York, 1916.  
THERMOCHEMISTRY. By Julius Thomsen, translated from the Danish by Katharine A. Burke. New York, 1908.

**Geology and Mineral Resources**

- CHILE. Published by the Chilean Government, 1915. (Gift of Arthur T. Ward.)  
CLAY DEPOSITS NEAR MOUNTAIN GLEN, UNION COUNTY, ILL. Extract from Bull. No. 36, Illinois State Geological Survey, Urbana, 1917.  
GEOLOGY OF GRAHAM ISLAND. British Columbia. Memoir 88. Canada Department of Mines, Geological Survey, Ottawa, 1916.  
PRACTICAL NOTES ON HYDROGRAPHIC AND MINING SURVEYS. By W. H. Hearing, Milwaukee, 1872.  
PROFILE SURVEYS IN 1915 ALONG THE RIO GRANDE, PECOS RIVER, AND MORA RIVER, NEW MEXICO. Water Supply Paper 421, U. S. Geological Survey, Washington, 1916.

- PROFILE SURVEYS ALONG HENRYS FORK, IDAHO, AND LOGAN RIVER AND BLACKSMITH FORK, UTAH. Water Supply Paper 420, U. S. Geological Survey, Washington, 1916.
- MINERAL RESOURCES OF MICHIGAN WITH STATISTICAL TABLES OF PRODUCTION AND VALUE OF MINERAL PRODUCTS FOR 1915 AND PRIOR YEARS. Publication No. 21, Geol. Ser. 17, Michigan Geological and Biological Survey, Lansing, 1916.
- USEFUL MINERALS OF THE UNITED STATES. Bull. No. 624, U. S. Geological Survey, Washington, 1917.
- YMIR MINING CAMP. British Columbia. Memoir 94, Canada. Department of Mines, Geological Survey, Ottawa, 1917.

### Gift of Carleton Greene

- COLLINS, JOHN. Explorations in Central America. (From Proceedings Royal Geographical Society of London, 1867.)
- COUVREUX, A. AND HERSENT, H. Les nouvelles installations maritimes du Port d'Anvers. Paris, 1879, 1880.
- DE ROUCHMONT, QUINETTE. Notice sur le port du Havre. Paris, 1875.
- NEW YORK BOARD OF WATER COMMISSIONERS. Brooklyn Water Works and Sewers. A descriptive memoir. New York, 1867.
- REYNAUD, LÉONCE. Memoir upon the illumination and beaconage of the coasts of France. Washington, 1876.
- U. S. DEPARTMENT OF STATE. Reports upon the survey of the boundary between the territory of the United States and the possessions of Great Britain from the Lake of the Woods to the Summit of the Rocky Mountains. Washington, 1878.
- . Joint maps of the northern boundary of the United States from the Lake of the Woods to the Summit of the Rocky Mountains.
- WILLIAMS, J. J. Isthmus of Tehuantepec, N. Y. 1852.

### General

- MINE ACCOUNTS AND MINING BOOK-KEEPING. Ed. 6. By James G. Lawn. London, Chas. Griffin & Co., 1909.
- SIMPLE MINE ACCOUNTING. By David Wallace. Ed. 2. New York, McGraw-Hill Book Company, 1909.
- UNIFIED ACCOUNTING METHODS FOR INDUSTRIALS. By C. E. Woods. New York, 1917.
- HANDBOOK OF ENGINEERING MATHEMATICS. By W. E. Wynne and Wm. Spratagen. New York, 1916.
- LUBRICATING ENGINEER'S HANDBOOK. By J. R. Battle. J. B. Lippincott, Philadelphia, 1916.
- ELECTRIC HEATING. By E. A. Wilcox. San Francisco, 1916.
- MECHANICAL HANDLING AND STORING OF MATERIAL. By George F. Zimmer. New York, 1916.
- OXY ACETYLENE WELDING. By S. W. Miller. Industrial Press, New York, 1916.
- CHARACTERISTICS OF UNIFLOW ENGINE GENERATING UNITS. By E. Hagenlocher. Reprint from Electrical World, Feb. 10, 1917. (Gift of Author.)
- LARGE ELECTRIC POWER STATIONS, THEIR DESIGN AND CONSTRUCTION. English translation. By G. Klingenberg. New York, 1916.
- STEAM BOILERS; THEIR THEORY AND DESIGN. By H. de B. Parsons. Ed. 5. New York, 1917.
- GREAT BRITAIN. Advisory Committee for Aeronautics. Technical Report, 1912-13. London, 1914.
- INTERNATIONAL MILITARY DIGEST. Annual 1916. New York, 1917.
- NATIONAL ADVISORY COMMITTEE FOR AERONAUTICS. Annual Report. Second, 1916. Washington, 1917. (Gift of Committee.)
- SUBMARINE TORPEDO BOAT. By Allen Hoar. New York, 1916.
- COMPULSORY HEALTH INSURANCE, SOME VITAL FACTS AND CONSIDERATIONS IN RESPECT TO. March, 1917. By M. W. Alexander. (Gift of author.)
- SAFETY REVIEW. American Smelting and Refining Co., New York, March, 1917.
- SUMMARY OF JUDGE KILLITS'S DECISION ON PEACEFUL PICKETING IN THE HOME TELEPHONE COMPANY CASE. n. p. n. d. (Gift of National Association of Manufacturers.)
- PENNSYLVANIA. Report of the Commission to Investigate the Increase in the Cost of Anthracite Coal in the Commonwealth of Pennsylvania. To the Governor and Legislature, pursuant to Joint Resolution of June 15, 1915.



**PLANNING PUBLIC EXPENDITURES TO COMPENSATE FOR DECREASED PRIVATE EMPLOYMENT DURING BUSINESS DEPRESSIONS.** Extension of an Address by John R. Shillady. Nov., 1916. (Gift of Mayor's Committee on Unemployment.)  
**PUBLIC OFFICE.** From a Civil Service and Business Point of View. By John J. Hopper. Albany, 1917. (Gift of Register of New York County.)  
**SPANISH.** Commercial and Professional. By Teodoro S. Romero. Philadelphia, 1916.

### Company Reports

**BROKEN HILL PROPRIETARY COMPANY, LTD.** Reports and Statements of account for half-year ending Nov. 30, 1916. Melbourne, 1916.  
Report of 63d half-yearly ordinary general meeting Feb. 23, 1917. (Gift of Company.)  
**GOLDFIELD CONSOLIDATED MINES COMPANY.** Annual Report 10th, 1916. (Gift of Company.)  
**NEVADA CONSOLIDATED COPPER COMPANY.** Annual Report, 1916. New York, 1916. (Gift of Company.)

### Trade Catalogs

**BRADY FOUNDRY COMPANY, JAMES A.** Chicago, Ill.  
Bull. No. 1. Cox-Fulton Water Cooled Mechanical Stoker.  
Bull. No. 2. Simplex Traveling Grate Stoker. 1916.  
Bull. No. 3. Brady Steam Jet Ash Conveyor. 1917.  
**DEISTER CONCENTRATOR COMPANY.** Fort Wayne, Ind. "The Table with the Capacity and Efficiency." Feb., 1917.  
**DIAMOND POWER SPECIALTY COMPANY.** Detroit, Mich. Power Notes. March, 1917.  
**FOUNDATION COMPANY.** New York, N. Y. Industrial Plants. Descriptive Pamphlet.  
**INGERSOLL-RAND COMPANY.** New York, N. Y.  
Form No. 3311.  
"Imperial X" Duplex Steam Driven Compressors. Ed. 5. Feb., 1917.  
Form No. 8507. "Little David" Pneumatic Drills. Feb., 1917.  
**SUPPLER-BIDDLE HARDWARE COMPANY.** Philadelphia, Pa. Monel Metal. April, 1917.  
**THWING INSTRUMENT COMPANY.** Philadelphia, Pa. Pyrometers for Clay Burners.

## MEMBERSHIP

## NEW MEMBERS

The following list comprises the names of those persons who became members during the period of Apr. 10, 1917, to May 10, 1917.

- ANDERSON, GEORGE K., JR., Supt., Iron Ore Mines, The Low Moor Iron Co. of Va.,  
Longdale, Va.
- AVERY, DAVID, Met.,..... Collins House, Collins St., Melbourne, Vic., Australia.
- BATES, WILLIAM T., Shift Foreman, American Zinc Co. of Tenn., Mascot, Tenn.
- BIRD, ELMER, Min. & Civil Engr.,..... P. O. Box 1483, Boise, Idaho.
- BLACKBURN, JOSEPH R., Vice-Pres. & Genl. Mgr., Austen Coal & Coke Co.,  
Austen, Preston Co., W. Va.
- BOTSFORD, FRANK P., Asst. Genl. Supt., Eastern Mesaba, Pickands, Mather & Co.,  
Gilbert, St. Louis Co., Minn.
- BRADEN, SPRUILLE, Min. Engr., Andes Exploration Co. of Maine, Casilla 83-D,  
Santiago, Chile, South America.
- BROWNE, DAVID G., Mill Designer, New Jersey Zinc Co., 55 Wall St.,  
New York, N. Y.
- BRUDERLIN, EMIL J., Met. Engr., Ohio & Colorado Smelt. & Ref. Co., Salida, Colo.
- CAMPBELL, DONALD C., Min. Engr., Campbell Wells & Elmendorf,  
114 James St., Seattle, Wash.
- CARRIGAN, J. J., Asst. Genl. Supt. of Mines, Anaconda Copper Min. Co.,  
Butte, Mont.
- CARROLL, ALEXANDER W., General Foreman, Raritan Copper Works, Perth Amboy,  
N. J.
- CASTLE, S. N., Commercial Engr., General Electric Co., 120 Broadway,  
New York, N. Y.
- CLEMMITT, WILLIS B.,..... Met. Engr., Bartlett-Hayward Co., Baltimore, Md.
- COOK, W. LORRAIN, Min. Engr., Examining Nitrate Properties, Casilla 810,  
Antofagasta, Chile, South America.
- COOLBAUGH, JOHN, Construction, 211 Penn Counties Trust Bldg., Allentown, Pa.
- CORBETT, CLIFTON S., Geologist, E. J. Longyear Co., 710 Security Bldg.,  
Minneapolis, Minn.
- DAVIDSON, LYNDALE P., Testing Dept., Anaconda Copper Min. Co., Anaconda, Mont.
- DAVIS, EDWARD W., Experimental Mining Engineering, with W. G. Swart,  
808 Sellwood Bldg., Duluth, Minn.
- DUDLEY, G. H., Mine Foreman, Phelps Dodge Corp., Morenci Branch,  
P. O. Box 937, Morenci, Ariz.
- EATON, WALTER J.,..... Asst. Supt., Tungsten Mines Co., Bishop, Cal.
- EDDINGFIELD, FRANK T., Mgr., Blanton Copper Mining Syndicate, Santo Domingo,  
Dominican Republic.
- ESTILL, HOWARD S., Chief Engr., Elk Horn Coal Corp., Wayland, Floyd Co., Ky.
- EVANS, WALTER F., Min. Engr., The New Jersey Zinc Co. (of Pa.)  
R. F. D. No. 1, So. Bethlehem, Pa.
- FIELDS, HOWARD H., Min. Engr., Genl. Mgr., Triangle Mining Co.,  
Salt Lake City, Utah.
- HELLER, IRA S., Chief Draughtsman, Concentrating Mills, The New Jersey Zinc Co.,  
55 Wall St., New York, N. Y.
- HORNER, ROY R., Min. Engr., U. S. Bureau of Mines, Univ. of Utah,  
Salt Lake City, Utah.
- HOUTEN, JAN VAN, Pres., St. Louis, Rocky Mt., and Pacific Co., Raton, N. Mex.
- HUTCHISON, L. L.,..... Pres., Kanola Oil Co., Box 846, Tulsa, Okla.
- ITO, ICHIRO, Supt., Osaka Seirenjo, Mitsubishi Goshi-Kaisha,  
Mitsubishi Osaka Seirenjo, Kitaku, Osaka, Japan.
- JONES, JAMES D.,..... Chief Engr., Indiana Steel Co., Gary, Ind.
- KIRK, CHARLES T., State Geologist, Professor of Geology, Univ. of New Mexico,  
Albuquerque, N. Mex.
- KOEBIG, JULIUS, Consulting Chemical and Mining Engineer, 329 Union Oil Bldg.,  
Los Angeles, Cal.
- LEHMANN, CHARLES, Asst. Mgr., Société des Mines de Cuivre de Naltagua,  
El Monte, Provincia de Santiago, Chile, South America.
- McCRARY, EDGAR W., Geologist,..... Box 1447, Tulsa, Okla.

- McCURDY, ELLIOTT S., Min. Engr., Utica Mining Co. & Treasure Mining Co.,  
576 Mills Bldg., San Francisco, Cal.
- MEISTER, OTTO F. .... Pres., Collinsville Zinc Co., Collinsville, Ill.
- METZ, GILBERT P. .... Plant Engr., The Atlas Portland Cement Co., Hannibal, Mo.
- NOEHL, B. F. .... Supt., Billie Mack Mining Co., Parker, Ariz.
- PARSONS, GEORGE H. .... Chief Chemist, Ray Cons. Copper Co., Hayden, Ariz.
- PEARSON, ALFRED, JR. .... Division Engr., Consolidation Coal Co., Somerset, Pa.
- PEW, JOHN G., Pres., The Peoples Natural Gas Co., 422 Sixth Ave., Pittsburgh, Pa.
- PICKARD, THOMAS R., Testing Dept., Anaconda Copper Min. Co.,  
P. O. Box 391, Anaconda, Mont.
- PLUMB, CARLTON H., Mining Engr. .... 313 Miners Bank Bldg., Joplin, Mo.
- PLUMMER, PHILIP J. A., Smelter Supt., Broken Hill Associated Smelters Prop., Ltd.,  
Port Pirie, South Australia.
- POLHEMUS, THEODORUS ... Supt., Langhorn Mine, Wisconsin Zinc Co., Benton, Wis.
- REA, WALTER C. .... Chile Exploration Co., Chuquicamata, Chile, South America.
- ROBINSON, CHARLES, Foreman of Smelters, Hampden Cloncurry Copper Mines,  
Ltd., Hampden Mines, Kuridala, North Queensland.
- RUDER, W. E., Met., General Electric Co., 642 Rugby Road, Schenectady, N. Y.
- SHANKLAND, A. D., Chief Chemist, Saucon Plant, Bethlehem Steel Co.,  
So. Bethlehem, Pa.
- SMITH, CHARLES A. .... Stope Engr., Calumet & Arizona Min. Co., Warren, Ariz.
- SMITH, H. G., Supt., La Luz Mine, La Luz & Los Angeles Min. Co.,  
Principulca, Nicaragua, Central America.
- SPOOR, HARRY M., Supt. of Mine, Bethlehem Chile Iron Mines Co., Minas Tofo,  
La Higuera, Coquimbo, Chile, South America.
- STADER, JAMES A. .... Engr., The Eagle-Picher Lead Co., Joplin, Mo.
- STARR, GEORGE W. .... Min. Engr., Empire Mines, Grass Valley, Cal.
- SUMAN, JOHN R., Asst. Geologist and Chief Engr., Rio Bravo Oil Co.,  
Room 716, Southern Pacific Bldg., Houston, Tex.
- SWEETSER, N. W., Supt. for Granby Cons. Mining, Smelting & Power Co.,  
Hadley, Alaska.
- SYKES, WESLEY P.; Met., Cleveland Wire Div. of National Lamp Works of  
The General Electric Co., 45th St. & Hough Ave., Cleveland, Ohio.
- TAYLOR, HOLLINSHEAD N., Pres., N. & G. Taylor Co., Inc., 300 Chestnut St.,  
Philadelphia, Pa.
- THOMAS, WILLIAM B., Acting Asst. Supt., Cia. Metalurgica de Torreon, S. A.,  
Apartado 93, Torreon, Coah., Mexico.
- THORNBURG, CHESLEY C., Civil Engr., The Spanish American Iron Co., Felton,  
Oriente, Cuba.
- TIPPETT, JOHN M., Asst. Supt., Milling Dept., Portland Gold Min. Co.,  
Colorado Springs, Col.
- TREICHLER, H. E., Asst. Mine Supt., Chile Exploration Co., Chuquicamata,  
Chile, South America.
- VARNEY, FRED W. .... Mining Engr., 350 E. So. Temple St., Salt Lake City, Utah.
- WAGNER, CONSTANTIN, Genl. Mgr., Guayacan Smelter, Ore Trading Co., Ltd.,  
Dept. Fundicion de Guayacan, Casilla 226, Coquimbo, Chile, South America.
- WEBB, KEITH P., Met., Electrolytic Mine Co. of Australasia, Risdon, Hobart,  
Tasmania.
- WENDEL, EDMUND, Testing Dept., Electric Railway Improvement Co.,  
Cleveland, Ohio.
- WHALEY, GEORGE C., Draftsman, St. Louis Smelt. & Ref. Co.,  
P. O. Box 227, St. Francois, Mo.
- WILLIAMS, EDWARD I., Asst. to Mgr. Min. and Met. Dept., General Chemical Co.,  
25 Broad St., New York, N. Y.
- WILLISTON, GEORGE F. .... Min. Engr., Nevada Arizona Mines Co., Hockberry, Ariz.
- YASUKAWA, SHOICHI, Min. Engr., Mitsubishi Goshi Kaisha, Sado Gold Mine,  
Aikawa, Sado, Japan.

### Associate Member

- KING, FRANK E., Engr. and Chem., Pres., Providence Brick Co., Providence, R. I.

### Junior Members

- BOYLES, R. C. .... Student, Pennsylvania State College, State College, Pa.
- BRAY, HARRISON M., Student, Pennsylvania State College, O. E. House,  
State College, Pa.

### CHANGE OF STATUS

### Associate to Member

### Junior Member to Member

**Total Membership, May 10, 1917..... 6,259**

## CANDIDATES FOR MEMBERSHIP

**APPLICATION FOR MEMBERSHIP.**—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

### ***Applications Lacking Endorsement***

Applications for membership have been received from Mr. Drewitt, Mr. Mackay and Mr. Trench, whose records are given below. These applications lack the necessary number of endorsers, but since these candidates live at some distance from the headquarters of the Institute, their records are published here in order that any members who are acquainted with them may be advised of the circumstances and may have an opportunity of writing to the Secretary endorsing these candidates.

**George Edward Drewitt, Mt. Morgan, Queensland, Aust.**

Proposed by A. A. Boyd.

Born 1895-1900, Higher Grade School, Gillingham, England. 1903-09, Electrical and Mechanical Engrg., Gillingham Tech. College (evening). 1909-10, Min. Engrg. course, South Kensington, London, England. 1900-09, Engr. and draftsman, Medway Iron Works, Rochester, England. 1910, Engr., Maschiner-patrick, Lubbeck, Germany. 1911-13, Asst., West African Mines. 1913, Reorgan-izing cement works and coal mines and reporting on properties, The Victorian Portland Cement Co., Richmond, Vict., Aust. 1912-13, Reporting, Pacific Phosphate Co., Ocean Island, Gilbert Group, Central Pacific. 1913-14, Reporting on mines and mills and supt. erection of mills.

**Present position: Engrg. Dept., Mt. Morgan Gold Mines Co., Ltd.**

**Alexander Dudley Mackay, Waratah, Tasmania.**

Proposed by J. D. Millen.

Born 1888, St. Lawrence, Queensland. 1903-06, Launceston Grammar School. 1907-09, University of Tasmania, B. Sc. 1910-12, University of Melbourne, B. M. E. 1916, Degree of M. M. E. 1912-13, Asst. to chief mining inspector, Mines Dept. of Victoria. 1913-16, Chief Assayer, engaged in research work, Mt. Bischoff Tin Mining Co.

Present position: Chief Assayer, Mt. Bischoff Tin Min. Co.

**Robert Hamilton Trench, Rangoon, Burma, India.**

Proposed by A. W. G. Bleeck, John Cadman.

Born 1880, Liverpool. 1902, Univ. of Cambridge, England, 2d class Mechanical Science. 1902-04, British Westinghouse Electric & Mfg. Co., Trafford Park, Manchester. 1904-08, Asst. Mgr. and Mgr., London & Pacific Petroleum Co. 1908-12, general reporting and executive work on properties in Russia, West Indies, East Indies, Salina, United States, Messrs. Thompson and Hunter, London.

Present position—1912 to date: Genl. Mgr., British Burmah Petroleum Co.

The following persons have been proposed during the period Apr. 10, 1917, to May 10, 1917, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

### Members

**Hirashi Abey**, Osarizawa Mine, Akitaken, Japan.

Proposed by S. Harada, T. Kawamura, M. Yamashita.

Born 1883, Japan. 1907, Graduated from the Mining and Met. College of Tokyo Imperial University, Japan. 1907, Mitsubishi Co., Japan.

Present position: Sub-Mgr., Osarizawa Mine.

**J. Carson Adkerson**, Woodstock, Va.

Proposed by J. H. Watkins, C. A. Burdick, T. L. Watson.

Born 1892, Lynchburg, Va. Feb., 1912, Graduate, Daniels High School, Lynchburg, Va. Feb., 1912 to Jan., 1913, Spent in doing general survey work, mining property, studying and serving as apprentice in mining under Dr. C. T. Hennig, N. Y. C. June, 1912 to March, 1914, Engineering and administrative work, Piedmont Manganeese Corp., Six Mile Bridge, Va., near Lynchburg, Va. March, 1914 to April, 1915, Examining and dealing in ores and properties. April, 1915 to Feb., 1916, Engr. and Mgr., Powells Fort Manganeese Mine, Woodstock, Va.

Present position—Feb., 1916 to date: Mgr., Stockwood Realty Corp. Mining Manganeese.

**John H. Ballamy**, Forty Fort, Pa.

Proposed by H. G. Davis, N. N. Nichols, S. D. Dimmick.

Born 1886, Plymouth, Pa. 1903, Graduated Plymouth, Pa., High School. 1904, Practical work as a miner with evening study in International Correspondence School Course in Civil Engineering. Home study in subsequent years tending toward qualifications as Mining Engineer. 1903, Commenced work in Coal Dept. as practical miner, Delaware, Lackawanna & Western R. R. Co. 1904, Commenced work on mine survey corps with same corporation. 1906, Became chief of mine survey party. 1908, Took up duties of map draftsman and general mining engineering office work. 1916, Became District Engineer in charge of mining engineering work. All the above for the Delaware, Lackawanna & Western R. R. Co.

Present position: District Engineer, Coal Dept., D. L. & W. R. R. Co.

**Leslie Park Barrett**, Lansing, Mich.

Proposed by C. K. Leith, W. J. Mead, W. E. Hopper.

Born 1887, Boston, Mass. June, 1907, Graduated St. Albans, Vermont, High School. Feb., 1913, Graduated University of Michigan, B. S. Degree. 1911-12, Employed by F. I. Carpenter Syndicate. Mar., 1913 to July, 1913, E. T. Longyear Co. August, 1913 to date, Michigan Geological Survey.

Present position: Asst. Geologist, Michigan Geological Survey.

**John Willis Bennie**, Clifton, Ariz.

Proposed by John D. Wanvig, Jr., R. B. Earling, John E. Penberthy.

Born, 1866, Glasgow, Scotland. 1881-84, Falkirk School of Science, West of Scotland Technical, Scotland. 1886-87, Finsbury Technical College. 1889-93, Chem. and Chief Chem., Rio Tinto Mining Co., Spain. 1892, Associate Member, Min. & Met., London. 1888-93, Rio Tinto Mines, Ltd., Spain. 1897-1901, Mountain Copper Co., California, Special Chem. and Supt. of Smelting.

Present position—1902 to date: Supt., now Genl. Mgr., Shannon Copper Co.

**Gunnar Bergström**, Stockholm, Sweden.

Proposed by Frank A. Ray, J. P. Hutchins, Ross B. Hoffmann.

Born, 1889, Falun, Sweden. 1908, Matriculation examination, Falun. 1912, Grad., School of Mines, Stockholm, Min. Engr. 1912-13, Asst. Geol., Geological Survey, Sweden, and Asst. at School of Mines, Stockholm. 1913, 4 months, Teacher, Mining High School, Falun. 1913-14, Min. Engr., Stripa Iron Mines, Sweden.

Present position—1914 to date: Chief Engr., Swedish Diamond Rock Drilling Co., Stockholm.

**Alexander J. Blackstone**, Lead, So. Dak.

Proposed by B. C. Yates, A. J. M. Ross, E. A. Brooks.

Born 1875, Cheyenne, Wyo. 1889-92, St. Johns Military School, Manlius, N. Y. 1892-96, Rensselaer Polytechnic Institute. 1896 to date, Homestake Mining Co., Timekeeper, Assayer, Surveyor.

Present position: Asst. Engr., Homestake Mining Co.

**Charles E. Bogardus**, Seattle, Wash.

Proposed by C. F. Lee, C. Simenstad, P. E. Wright.

Born 1863, Champaign, Ill. 1883, University of Ill., B. S. Chemistry. 1887, Employed by Mariner & Hoskins, Chicago. 1889, B. & O. Ry. as chemist. 1890, Asst. Professor Chemistry, University of Ill. 1890 to present time, Have conducted a commercial laboratory as assayer and chemist in Seattle, Wash.

Present position: Assayer and Chemist.

**Owen Fugate Brinton**, Quapaw, Okla.

Proposed by C. P. Perin, C. H. Bowman, T. Simons.

Born 1884, Council Bluffs, Iowa. 1901, Graduate Butte High School. 1905, Graduate Montana State School of Mines, degree of Engineer of Mines. 1905-06, Anaconda Copper Co., Butte, Mont., as Miner, Timekeeper and Sampler. 1906-07, Alexander Burrell, Edgemont, Nev., as Timekeeper and Engineer. 1907-08, Timekeeper, International Smelting & Ref. Co., Tooele, Utah. 1908-09, Asst. Supt., Fairview Fluorspar & Lead Co., Fairview, Ill. 1909-10, Asst. Supt., Rosiclare Lead Co., Rosiclare, Ill. 1910-11, Mgr., Arizona Alpha Min. Co., Kingman, Ariz. 1911-1913, Supt., Gold Road Mines Co., Gold Road, Ariz. 1913-14, Engr. on drill tests, Anaconda Copper Co., Butte, Mont. 1914-16, Supt. Tiger Tunnel, Burma Mines, Ltd., Nanttu, Burma.

Present position: Genl. Mgr., Admiralty Zinc Co.

**Alan Bruyere**, Tulsa, Okla.

Proposed by E. G. Woodruff, W. E. Pratt, E. W. Parker.

Born 1888, Larimore, N. Dak. 1902-06, High School, Larimore, N. Dak. 1907-1911, University of N. Dak., Degree E. M. in 1911. 1911, Employed by the U. S. Geological Survey. 1912, Employed by E. L. De Golyer, Eagle Oil Co. of Mex. 1913-14, John A. Bohland, Bridge Engr., Great Northern Railway, St. Paul. 1915-1916, E. G. Woodruff, Chief Geologist, Producers Oil Co., Houston, Tex.

Present position: Geologist with Producers Oil Co.

**Leonard Jerome Buck**, Bethlehem, Pa.

Proposed by C. A. Buck, H. Eckfeldt, J. W. Richards.

Born 1893, Bethlehem, Pa. 1915, E. M. Degree, Lehigh University. 1915-17, Lehigh Coke Co., Bethlehem, Pa.

Present position: Post Graduate Student, Lehigh University.

**William A. Buddecke**, Mineral Point, Mo.

Proposed by R. W. Barrell, J. D. Robertson, W. E. McCourt.

Born 1870, St. Louis, Mo., General Education Public Schools and Smith Academy W. U. 1900-02, Manville Covering Co. 1902-05, Mgr., H. W. Johns-Manville Co., St. Louis Branch. 1900-08, Pres. Los Vegas Ry. & Power Co., East Los Vegas, N. Mex. 1903 to date, Pres., Point Milling & Mfg. Co.

Present position: Pres., Point Milling & Mfg. Co.

**A. J. Burnham**, Webb City, Mo.

Proposed by J. N. Houser, H. I. Young, M. H. Newman.

Born 1872, Chicago, Ill. Common Public Schools in Chicago. 1886-88, Chicago Manual Training School. 1889-91, Secy., Lick Observatory, San Jose, Cal. 1891-99, Coffee planter, Miner in Central America. 1899-1902, U. S. Courts, Chicago. 1903-06, Steel business, Pittsburgh. 1906-17, Mine mgr. and owner in Joplin District.

Present position: Mine owner and operator.

**Allison Butts**, South Bethlehem, Pa.

Proposed by J. W. Richards, G. A. Roush, S. Rolle.

Born 1890, Poughkeepsie, N. Y. 1908-11, Princeton University, A. B. 1912-13, Mass. Inst. Technology, S. B. in Mining. 1913, Smelter sub-foreman, United States Metals Refining Co., Chrome, N. J. 1914-16, Experimental Chemist, U. S. Metals Refining Co., Chrome, N. J.

Present position: Asst. Editor, *The Mineral Industry*.

**Albert L. Coolidge**, Lead, So. Dak.

Proposed by B. C. Yates, R. Blackstone, E. F. Irwin.

Born 1875, Columbus, Neb. 1893, Graduate, Columbus, Neb., High School. Partial Course Metal Mining I. C. S. 1897-02, Homestake Mining Co., quartz millman. 1902 to date, surveyor.

Present position: Civil Engr., Homestake Mining Co.

**Ralph Emerson Davis**, Platteville, Wis.

Proposed by W. A. Smith, H. A. Buehler, W. E. McCourt.

Born 1884, Blacksburg, Va. 1902-06, University of Wisconsin, B. S. in Civil Engrg. 1906, Vinegar Hill Zinc Co. 1907-09, Chief Engr., Pittsburgh & Montana Copper Co., Butte, Mont. 1909-10, Instructor, Montana State School of Mines. 1910-11, Field Engr., Globe Exploration Co., Butte, Mont. 1911-16, Director, Wisconsin Mining School, Platteville, Wis.

Present position: Director, Wis. Mining School.

**John Halifax Dietz**, Fort Smith, Ark.

Proposed by R. G. Hall, H. Denman, H. O. Lewis.

Born 1873, Caro, Mich. 1892-94, University of Mich., Civil Engrg. 1896, Engr., Kansas & Texas Coal Co., St. Louis, Mo. 1899-1902, Engr., Central Coal & Coke Co., Kansas City, Mo. 1903, Engr., United Iron Works Co., Springfield, Mo. 1905, Supt., United Iron Works Co., Springfield, Mo. 1906-07, Cons. Engr. for the same company. 1907-09, Pres., Eagle Foundry & Machine Co., Ft. Scott, Kansas. 1909-10, Min. Engr., English Iron Works Co., Kansas City, Mo. 1910-14, Cons. Engr., Cleveland, Ohio. 1914-16, Cons. Engr., Fort Smith, Ark.

Present position: Cons. Engr., Mining, Zinc Smelting & Factory Construction.

**John E. Donahoe**, Joplin, Mo.

Proposed by E. F. Woodson, B. L. Wolfe, W. E. McCourt.

Born 1895, Granby, Mo. 1913, Graduate Joplin High School. Half year Christian Bros. College, St. Louis, Mo. Two and one-half years Kansas State School of Mines. Experience in mill-work, mapping (surface and mine), mill-man for Granby Min. & Smelt. Co. (Now American Zinc, Lead and Smelting Co.) Asst. flotation operator for Denver Min. & Flotation Co., Joplin, Mo.

Present position: Supt. of Flotation for Oronogo Circle Mining Co., Oronogo, Mo.

**Charles H. Dunning**, Prescott, Ariz.

Proposed by E. S. Smith, J. N. D. Gray, D. H. Bradley, Jr.

Born 1887, Auburn, N. Y. 1909, Yale University, Min. Engineering course, Degree Ph. B. 1908, Asst. Engr., Exploration work, Globe Mine Exploration Co., Idaho. 1909, Asst. Engr., Examination work, Northern Ontario Exploration Syndicate, Cobalt District, Underground experience, Cobalt. 1910, Asst. Supt., Big Pine Mining Co., Ariz. 1911 to date, Mgr., Big Pine Cons. Mining Co., Ariz.

Present position: Mgr., Big Pine Cons. Mining Co.

**Francis Bird Dutton**, New York, N. Y.

Proposed by J. V. W. Reynders, D. Dows, C. T. Rand.

Born 1875, Northboro, Mass. 1897, A. B., Harvard College. 1900, S. B., Mass. Inst. Tech. 1900, Great Northboro Paper Co. 1900, The Penna. Steel Co., Steelton & Lebanon, Pa., as Mgr. of Lebanon Plant. 1914, Mgr. of Mayari Mines, Spanish American Iron Co., Felton, Cuba. 1915, The Penna. Steel Co., Mgr. of Lebanon, Pa., Plant, later Bethlehem Steel Co. 1917, Genl. Supt., Wharton Steel Co. 1917, American Grondal Co.

Present position: Mining Engineer, American Grondal Co.

**George Richard Fausett**, Tucson, Ariz.

Proposed by C. F. Willis, C. A. Mitke, J. P. Hodgson.

Born 1884, New Haven, Conn. 1902-05, Sheffield Scientific School, Ph. B. 1907-08, University of Washington, School of Mines, mineralogy and assaying. 1915-16, Columbia School of Mines, flotation and ore dressing. 1905-06, Draftsman, The

**National Pipe Bending Co.** 1906-07, Instrument man, Construction on The Brighton Beach Improvement. 1907-08, Special work at the University of Washington School of Mines. 1908-09, Instrument man, The Grand Trunk Pacific R. R., Prince Rupert, B. C. 1912-15, Min. Engr., Cerro de Pasco Mining Co., Cerro de Pasco, Peru. Present position: Engr., Arizona State Bureau of Mines.

**Abraham Leonardo Fernandez**, El Paso, Tex.

Proposed by A. L. McRae, H. T. Mann, Chas. Y. Clayton.

Born 1878, Monterrey, Mexico. 1900, Grad., B. S., Mining Engineering, Missouri School of Mines. 1902, E. M., Missouri School of Mines. 1900-02, Asst. Assayer, Asst. Engr., Mexican Lead Co. of Monterrey, Mexico, transferred to Montezuma Lead Co. of Sta. Barbara, Chih., as Assayer and Chem. 1903, Engr., A. Longega, Guanacevi, Durango, Mexico. 1904, Chief Engr., S. Min. Dept., Cia. Metalurgica de Torreon, S. A. 1905-07, Mgr. & Chief Engr., A. Longega, Guanacevi, Durango, Mexico, and Taxco Gro. 1907-09, Cons. Min. Engr., Monterrey, Mexico. 1909-10, Mgr., Bruni & Sanchez, La Portilla, Durango, Mexico. 1911-12, Mine Supt., Cia. Minera Ignacio Rodriguez Ramos, S. A., Almoloya, Chih. 1913-15, Cons. Min. Engr., Monterrey, Mexico.

Present position: Genl. Supt., Cia. Minera Ignacio Rodriguez Ramos, S. A., Almoloya, Chih., Mexico.

**Andrew Ellett Fleming**, Huntingdon, Pa.

Proposed by H. Eckfeldt, H. S. Drinker, B. L. Miller.

Born 1891, Richmond, Va. 1906-09, Richmond High School, Richmond, Va. 1911-13, Mining Engineering Course, Va. Poly. Inst., Blacksburg, Va. 1913-16, Lehigh University. 1909-11, Rodman, C. & O. Ry. Co. 1911-12-13-14-15-16, Employed by several W. Va. coal mines at various positions (practical mining and some engineering). 1916-17, Engineer, Loup Creek Colliery Co., Page, W. Va.

Present position: Mine Inspector, Pa. Bit. Mutual Ass'n.

**Victor Max Frey**, York, Pa.

Proposed by J. T. Singewald, Jr., W. B. Clark, W. A. Baker, Jr.

Born 1886, York, Pa. 1906, Johns Hopkins Univ., A. B. 1908, Mass. Inst. of Tech., B. S. 1909, La Belle Iron Works, Steubenville, Ohio. 1909-17, J. E. Baker Co., York, Pa. Lime, crushed and fluxing stones. This company operates six lime and stone plants. 1915-17, Director and Treasurer, Kaptom Coal Co., York, Pa.

Present position: Genl. Supt., J. E. Baker Co.

**Charles S. Galbraith**, Webb City, Mo.

Proposed by J. M. Callow, E. Juessen, C. E. Grunsky, Jr.

Born 1875, Kew, Vic., Aust. 1886-89, High School, Kew, Victoria, Aust. 1890-91, Scotch College, Melbourne, Victoria. 1892-97, The University of Melbourne. 1898-99, Engineering course in Civil, Mechanical, Mining and Metallurgy. Graduated with honors, won Dixon Scholarship in Mech., Min. and Met. and 1899, obtained B. C. E. Degree. 1914, Elected Assoc. Member Institution of Civil Engineers, London. 1915, Obtained degree of Master of Civil Engrg., Melbourne University. 1898, Foreman Cyanide Works, So. German G. M. Co., Maldon, Vic. 1900, Met., Lake George Mines, N. S. W. 1901, Constructional Draftsman, North Mt. Lyell Copper Co.'s Smelter, Crotty, Tasmania. 1902, Genl. Mgr., Mt. Welcome G. M. Co., Mitta Mitta, Vic. 1903-05, Chief Engr., N. Guthridge Ltd., Mining Machinery Manufacturers, Sydney, N. S. W. 1906-07, Mgr., De Bavay Zinc Treatment Co., Broken Hill, N. S. W. 1908, Constructional Engr., Mt. Bischoff Tin Mining Co., Waratah, Tasmania. 1909-12, Asst. Engr., in charge of Construction Harbor Trust, Sydney, N. S. W. 1912-15, Principal Asst. Engr., Harbor Trust, Melbourne, Vic. 1915, Asst. Engr., on Mill Construction Engels Coppér Min. Co., Plumas, Cal.

Present position: Rep. for J. M. Callow (Metals Recovery Co.) in the states Missouri and Oklahoma.

**Edward V. Graybeal**, Great Falls, Mont.

Proposed by E. P. Mathewson, F. J. Brulé, B. B. Thayer.

Born 1892, Jefferson, N. C. 1914, Graduated Colorado School of Mines, E. M. 1915 until present, Employed by Anaconda Copper Mining Co.

Present position: General Foreman, Zinc Plant, Anaconda Copper Mining Co.

**Albert W. Hahn**, New York, N. Y.

Proposed by R. M. Raymond, E. J. Hall, E. B. Kirby.

Born 1882, New York, N. Y. 1890-97, New York Public Schools. 1897-1901, High School. 1901-05, Columbia Univ., B. S. 1905-06, Chem., Glen Flora Sugar



Co. 1906, Partner, C. W. Carpenter & Co., Custom Assayers, Mexico City, Mex. 1907, Chem. O. & T. Braniff, Mexico City, Mex. 1907-08, Chem., Mexican National Sugar Refining Co., Mexico. 1908-09, Mine Examination Work. 1909-11, Metallurgist, Reforma M. & M. Co. 1911-12, Mine Examination Work. 1913, Process Investigations, John Hays Hammond. 1914, Process investigations, Sta. Gertrudis Co., Ltd.

Present position—1915 to date: Office Manager and Secretary respectively, Electrolytic Zinc Co., Inc., and Metals Disintegrating Co., Inc.

**John S. Harris**, Granby, Mo.

Proposed by M. H. Newman, H. I. Young, A. W. Koch.

Born 1894, Ashland, Mo. General grade and high school training. University of Missouri, 1911-14. 1914-15, American Zinc Co. of Tenn. 1915, American Zinc, Lead and Smelt. Co., Cartersville, Mo.

Present position: Flotation Engr., American Zinc, Lead and Smelt. Co.

**Raymond William Hawley**, Porterville, Cal.

Proposed by J. Spotts McDowell, Lester C. Uren, W. S. Morley.

Born 1886, Hastings, Minn. 1900-04, High school, Worthington, Minn. 1908-10, Univ. of Minnesota, School of Mines. 1910-13, Univ. of California, B. S. 1913-15, One year Assistant Chemist, one year as Chief Chemist, Portland Gas & Coke Co., Portland, Ore. Various vacation periods spent in railway construction in North Dakota and Minnesota. Summer, 1911, Melones, Cal., underground work and lower house operator.

Present position—1915 to date: Shift boss, chemist, Porterville Magnesite Co.

**William J. Hayes**, Istmina, Republic of Colombia, S. Amer.

Proposed by N. C. Marshall, J. T. Dixon, H. G. Granger.

Born 1888, Omaha, Neb. St. Mary's College, Kansas. Creighton University, Omaha. 1911, Union Pacific Railroad, Omaha, Neb. 1912, Union Pacific Railroad, Rock River, Wyo. 1912, Union Pacific Railroad, Walcott, Wyo. 1913, Personal Account. 1915, Anglo-Colombian Development Co., Andagoya, Colombia, S. A. 1916, Compania Minera Chocó Pacifico, Istmina, Colombia, S. A.

Present position: Asst. Engr., Cia. Minera Chocó Pacifico.

**Fred J. Heckel**, Bradford, Pa.

Proposed by A. N. Diehl, B. R. Putnam, T. N. Barnsdall, 2d.

Born 1875, Buffalo, N. Y. 1899, B. S., Pennsylvania State College. 1911, M. S., Pennsylvania State College. 1901-03, Assayer, Montana Ore Purchasing Co., Butte, Mont. 1904, Assayer, Encampment Smelting Co., Grand Encampment, Wyo. 1907, Natural Gas Pipe Line Construction, Kansas. 1908-12, Asst. Supt., Lewis Rim Mfg. Co., Bradford, Pa. 1912-16, Manufacturing and Analytical Chemist, T. H. Quinn & Co., Olean, N. Y. 1916-17, Oil well, Petroleum Engineering.

Present position: Industrial, manufacturing and consulting chemist. Factories producing alcohol, charcoal, and acetate of lime.

**Albert A. Hoffman**, Chafaral, Chile, S. A.

Proposed by W. L. Du Moulin, A. K. Adams, W. Jenney.

Born 1880, Parsons, Kansas. 1905, School of Mines, Kansas University, B. S. 1905-07, Mill operator and Repair Foreman, Cananea Cons. Copper Co., Cananea. 1907-12, Master Mechanic, Braden Copper Co., Rancagua, Chile. 1912-15, Master Mechanic and Supt. Construction, Chile Exploration Co., Chuquicamata.

Present position—1916 to date: Supt. of Construction, Andes Copper Co.

**Paul F. Holstein**, Wilmington, Del.

Proposed by G. B. Street, H. G. Haskell, F. S. MacGregor.

Born 1888, Akron, Ohio. 1906-10, Case School of Applied Science, B. S. 1910, Eastern Laboratory, E. I. du Pont de Nemours Powder Co. 1911-17, Chief Chemist, Oficina Delaware, Taltal, Chile, of du Pont Nitrate Co.

Present position: Chemist, E. I. du Pont de Nemours & Co.

**Frederic Wilton James**, West New Brighton, S. I.

Proposed by J. F. Kemp, T. T. Read, G. F. Kunz.

Born 1880, Reading, England. To 1898, Trinity School, New York, N. Y. 1898-1900, School of Mines, Columbia Univ. 1900, Asst. Chem., New York Chemical Refining Co., Elizabeth, N. J. 1901-02, Draftsman, Gas Engine Work, Frederick Mfg. Co., Newark, N. J. 1902, Draftsman, Marine Boilers, Babcock & Wilcox. 1903-06, Chief Draftsman, H. Huenekes Co., New York, N. Y., Brick Manufacturing Plants. 1903-07, Draftsman, C. W. Hunt Co. 1907-10, Draftsman & Asst. Engr., Cement Engineering & Construction Co., Alsen, N. Y. 1907-10, Cement Engineering

Co., Hudson, N. Y. and Yorktown, Va. 1911-13, Engr., American Clay Machy. Co., Willoughby, O.

Present position—1913 to date: Engr., Surety Engineering Co., New York.

**Charles T. Joslin, Prescott, Ariz.**

Proposed by W. F. Staunton, J. L. Phillips, W. L. Clark.

Born 1863, Muskigon, Mich. 1878-80, Mining at Champion Iron Co. Mine, Beacon, Mich. and night study. 1881-82, Attended Lake Forest University but did not stay to graduate. 1882-83-84, Mining Iron at Negaunee and Lake Michigamme, Mich. 1885-86-87-88-89, In banking business, Marquette, Mich. 1890-1903, Banking business, Chicago; Cashier, West Chicago Bank; Asst. Sec'y and Treas., Title Guar. & Trust Co.; Cashier, Chicago Title & Trust Co. 1903, Opened up McCabe Mine. 1904, Yavapai Co., Ariz. Ran mine, closed mine when local smelter burned. 1906, Bought out machinery house of Brown Bros., Prescott, Ariz. Formed Ariz. Mine Supply Co.; have been Pres. ten years. 1917, Interested in mines in Crown King, Walker & Jerome.

Present position: Pres., Arizona Mines Supply Co.

**Edmund S. Leaver, Tucson, Ariz.**

Proposed by C. E. Van Barneveld, F. Anderson, A. K. Tiernan.

Born 1873, Salt Lake City, Utah. Salt Lake Public Schools. 1889-94, University of Utah. 1894-97, Met. at Marion and Gyser-Marion Mines, Mercur, Utah. 1897-1903, Various positions under Charles Butters. 1904-06, Supt., Dexter-Tuscarora Cons. Min. Co. under J. Dorn, Salt Lake City, Utah. 1906-09, Genl. Supt., Nevada-Goldfield Reduction Co. Works, Goldfield, Nev. 1910-12, Mgr., Daisy-Combination Mining Co. 1914-16, Supt., Knight Christensen Metallurgical Co. 1917, Metallurgist, U. S. Bureau of Mines.

Present position: Metallurgist, U. S. Bureau of Mines, Tuscon Experiment Station.

**Ernest Levy, Spokane, Wash.**

Proposed by L. K. Armstrong, J. McD. Porter, G. B. Dennis.

Born 1878; London, Eng. Till 1894, University College School, London, Eng. 1894-95, Falk Real Gymnasium, Berlin, Germany. 1895-96, Institution Ballitsee, Geneva, Switzerland. 1897-1901, Royal College of Science & Royal School of Mines, London, Eng. Associate Royal School of Mines. Member of Institution of Mining & Metallurgy. 1901-03, Asst. to A. Chester Beatty, Denver, Colo. 1903-05, Mgr., Le Roi No. 2 Ltd., Rossland, B. C. 1905-08, Asst. to Alexander, Hill & Stewart, London, Eng. 1908-09, Mgr., San Dionisio Mine of Rio Tinto Co., Spain.

Present position—1909 to date: Representative for Alexander Hill & Stewart, of London, in Canada and U. S. A.

**Whitney Lewis, Tulsa, Okla.**

Proposed by D. Hager, C. L. Severy, J. S. Irwin.

Born 1889, Louisville, Ky. June, 1906, Finished 3 years Western Military Academy (Class A school). June, 1912, Graduated (Engineer of Mines degree), University of Minnesota, Minneapolis, Minn. June, 1909, Miner with High Ore Mine, Butte, Mont. Sept.-Dec., 1909, Trammer with Michigan Utah Mine, Alta, Utah. Jan.-June, 1910, Chainman and Rodman, Oregon Short Line Railway. July-Sept., 1911, Miner, Mammoth Copper Co., Redding, Cal. June, 1912, Hoistman, Michigan Utah Mine, Alta, Utah. July-Oct., 1912, Hydrographer Price River Irrigation Co., Price, Utah. Jan.-Feb., 1913, Laboratory Asst., Smith Emory Co., Los Angeles, Cal. Feb., 1913-Sept., 1914, Geologist, Highway Engr. and Chief Ry. Location Engr., Carribean Pet. Co., Venezuela, South America.

Present position—1914 to date: Consulting Petroleum Geologist and Engineer.

**Harry A. Linch, Mogollon, N. Mexico.**

Proposed by S. J. Kidder, S. P. Hatchett, G. C. Baer.

Born 1888, Steubenville, Ohio. 1894-1906, Public Schools. 1906, Graduated from Steubenville High School. 1907-09, Ohio State University. 1911-13, Ohio State University. June, 1913, Graduated from Ohio State University, Degree Engr. of Mines. 1909-11, With Carnegie Steel Co., Mingo Junction, Ohio, in the Chemical laboratory and the Blast Furnace office. 1913-14, With Cleveland & Wetherhead as Solution man and Shift Boss in Gold & Silver Cyaniding Mill, Mogollon, New Mex. 1914-15, With Mogollon Mines Co. in Cyaniding Mill as Sand Plant helper, Filter man, Refinery man, Shift Boss. 1915-16, Mine sampler for the Mogollon Mines Co. 1916-17, Mine Shift Boss for the Mogollon Mines Co.

Present position: Mine Engineer, Mogollon Mines Co.

**A. Stafford McCullough, Tulsa, Okla.**

Proposed by D. Hager, C. L. Severy, J. S. Irwin.

Born 1885, Clifton, Ohio. 1905-08, Antioch College, Yellow Springs, Ohio. 1908-11, Stanford University, B. A. Degree, Cal. 1913, Petroleum Geologist with The Barber Asphalt Co., operating in Venezuela.

Present position—1916 to date: Geol., Carter Oil Co.

**Earl Joesting McNely, Alton, Ill.**

Proposed by Chas. Y. Clayton, H. A. Buehler, C. R. Forbes.

Born 1888, Alton, Ill. 1907-13, Shurtleff College, Alton, Ill., B. A. 1913-16, Missouri School of Mines, Rolla, Mo., B. S. 1916 to date, American Steel Foundries, Granite City, Ill.

Present position: Open Hearth Dept., American Steel Foundries, Granite City, Ill.

**Rodney A. Mercur, Jr., Roebbling, N. J.**

Proposed by C. G. Roebbling, W. Gummere, H. C. Boynton.

Born 1884, Towanda, Pa. 1891-1901, Towanda, Pa., Public Schools. 1901-03, St. Johns School, Manlius, N. Y. 1903-07, Lehigh Univ., Degree M. E. 1907-09, Youngstown Sheet & Tube Co., Eng. & Blast Furnace Depts. 1909-10, Bethlehem Steel Co., Open Hearth Dept. 1910 to date, Open Hearth Dept., Sales Dept., Engineering Dept., John A. Roebbling's Sons Co., Trenton, N. J.

Present position: Asst. Supt., Steel Mill.

**Sidney Norman, Spokane, Wash.**

Proposed by L. K. Armstrong, J. McD. Porter, J. C. Ralston.

Born 1870, Cheltenham, Eng. Educated at home and Modern School, Cheltenham, England, under late Richard Tyver. 1889, Arrived Spokane. 1893, Bryan Mining Experience with Slocan "boom." 1895-96, Roseland, B. C. 1896-1906, Spokane, Wash. 1906-11, Greenwater, Cal., Goldfield, Tonopah and other Nevada camps, Editor and Proprietor, *Los Angeles Mining Review*. 1912-17, Spokane, Wash.

Present position: Mine operator and proprietor, *Mining Truth*.

**Clarence LeRoy Olson, Rockland, Nev.**

Proposed by L. Smith, E. J. Schrader, C. F. Burt.

Born 1884, Wilson, Kansas. Chemical Engineering (B. S. in M. E.). 1907, Colorado Agricultural College, Fort Collins, Colo. Cripple Creek, Colo., Mining and milling previous to 1904, date of Graduation. 1907, Millman, Tonopah Belmont Development Co., Millers, Nev. 1907-10, Head Assayer and Chemist, Tonopah Mining Co., Millers, Nev. 1910-14, Mill Supt. and Genl. Mgr., Tonopah Liberty Mining Co., Tonopah, Nev. 1914-15, Millman and Shift Boss, Aurora Consolidated Mines Co., Aurora, Nev. 1915-17, Mill Supt., Pittsburg Dolores Mining Co., Yerington, Nev.

Present position: Mill Supt., P. D. M. Co.

**Edwin H. Peirce, Worcester, Mass.**

Proposed by H. M. Howe, A. Sauveur, H. M. Boylston.

Born 1881, Everett, Mass. 1896-1900, High School. 1900-04, Harvard University, S. B. 1906-07, Lowell Institute School for Industrial Foremen. 1904-06, U. S. Naval Torpedo Station as Chemist. 1906-07, Chemist, American Telegraph & Telephone Co. 1907 to date, American Steel & Wire Co.

Present position: Chief of Physical Laboratories, American Steel & Wire Co.

**William A. Perry, Pittsburgh, Pa.**

Proposed by John E. Perry, Geo. H. Morse, Trevor B. Simon.

Born, 1881, New England, O. 1887-1900, Common School. 1900-02, Ohio Univ., Elec. Engr. 1905-06, Ohio State Univ., Min. Engr. 1908-10, Ohio State Univ., Min. Engr. 1903, Morgan-Gardner Elec. Co. 1904-05, Forsythe Coal Co., Cambridge, O., Master Mech. 1906-08, Mexican Coal & Coke Co., Eng. Corps, Mine Supt. 1911-12, Mine Supt., Conn. Coke Region. 1912-13, Div. Supt., United Coal Co., Pittsburgh, Pa.

Present position—1913 to date: Genl. Mgr., Greensburg-Connellsville Coal & Coke Co.

**John Rothwell Pemberton, Tulsa, Okla.**

Proposed by C. L. Severy, J. S. Irwin, D. Hager.

Born 1884, Los Angeles, Cal. 1909-10, A. B., Stanford University. Instruction in Geology, Stanford University and advanced study for A. M. 1911-14, Geologist, Argentine Government, Exploratory work in Patagonia. 1915, With Ventura Refining Co., Los Angeles, Cal. 1908, Temporary appointment U. S. G. S. with Ralph

**Arnold, Geological Work in Cal.** 1917, Consulting Geologist, Tulsa, Okla.

Present position: Consulting Geologist.

**Fred S. Pfahler, Gillespie, Ill.**

Proposed by F. F. Jorgensen, W. L. Morgan, T. T. Brewster.

Born 1882, McGregor, Iowa. 1901, Graduated Mason City, Iowa, High School. 1904, Three Years, University of Wis. 1904-07, C. B. & S. Ry., Engrg. Dept. 1907-1908, Asst. Engr., Son. Coal, Coke & Mining Co., Belleville, Ill. 1908-10, Chief Engr., Son. Coal, Coke & Min. Co., Belleville, Ill. 1910-17, Genl. Supt., Son. Coal, Coke & Min. Co., Belleville, Ill.

Present position: Genl. Supt., C. & N. W. Coal Properties.

**Sidney Powers, Shreveport, La.**

Proposed by Wallace E. Pratt, W. E. Wrather, J. M. Lovejoy.

Born 1890, Troy, N. Y. 1911, B. A., Williams College. 1913, S. M., Mass. Institute of Technology. 1915, M. A. and Ph. D., Harvard Univ. 1914-15, Asst., Geology, Harvard Univ. 1915-16, Travelling Fellow (Sheldon Foundation) and Research Fellow, Harvard Univ. 1916-17, Division Geol., Producers Oil Co., Houston, Tex. Various publications, A. I. M. E. Transactions, Journal of Geology, American Journal of Science, Zeit. Valkomalogie, Geographical Review, Bulletin, Geographical Society of America, Bulletin, American Geographical Society, etc.

Present position: Div. Geol., Producers Oil Co.

**Alfred H. Ramage, Joplin, Mo.**

Proposed by D. M. Armistead, H. A. Buehler, P. N. Moore.

Born 1888, Oil City, Pa. 1908, Graduated in the Scientific Course from the Tome School, Port Deposit, Maryland. Attended Amherst College, Amherst, Mass., 3½ years. Connected with S. Y. Ramage in oil producing and zinc and lead mining since 1912. At present time manager of four zinc and lead mines.

Present position: Mgr., Zinc and Lead Mines.

**George Joseph Salmon, Cullasaja, N. C.**

Proposed by F. Bradshaw, F. W. MacLennan, J. H. Hensley, Jr.

Born 1891, Philadelphia, Pa. 1908, Graduated Central Manual Training High School, Philadelphia, Pa. 1910-14, Graduated Pa. State College, B. S. in Mining Engineering. 1908-10, Mineralogist, Foote Mineral Co., Philadelphia, Pa. Summers 1912-13, Mining, smelting and engineering work, North American Smelting Co., Kingston, Ont. 1914-15, Transitman, New River Collieries Co., Eccles, W. Va. June, 1915-July, 1916, Mining with Tonopah-Belmont Dev. Co., Tonopah, Nev.; Miami Copper Co., Miami, Ariz. as ore buyer, Foote Mineral Co., Philadelphia, Pa.

Present position: Min. Engr., for W. P. Leshure and Associates.

**Douglas Ramsay Semmes, Wichita Falls, Tex.**

Proposed by T. L. Watson, C. P. Berkey, J. F. Kemp.

Born 1892, Alexandria, Va. 1912, B. A., University of Va. 1913, M. A., University of Va. 1917, Ph. D., Columbia University, Economic Geology and Mining. Summer, 1915, Insular Govt. of Porto Rico, Natural History Survey of Porto Rico. 1916, Producers Oil Co., Geologist.

Present position: Geologist, Producers Oil Co.

**Warren Barton Shackelford, Webb City, Mo.**

Proposed by V. Rakowsky, H. A. Buehler, P. N. Moore.

Born 1862, Rockville, Ind. Received no technical education. Began active mining with other associates in 1897; old properties worked out or sold out. 1908-09, Was Supt. for the Good Mining Co., Webb City, Mo. 1909, Mgr., for S. Y. Ramage and continued with him until Feb. 1, 1916. 1916, Genl. Mgr., for Aiken, Curtis & Cosden and am associated with them in various mining interests and still hold my title.

Present position: Genl. Mgr., Aiken, Curtis & Cosden Interests.

**Nicholas C. Sheridan, Sunset, Idaho.**

Proposed by M. F. Quinn, W. H. Linney, L. K. Armstrong.

Born 1877, San Francisco, Cal. 1906, University of Idaho, B. S. in M. E. June to July, 1906, Millman, Hercules Mining Co. July, 1906, Assayer, Gold Hunter Min. Co. July, 1906-Feb., 1910, Min. Engr., Snow Storm Min. Co. Feb. to April, 1910, Expert, Bunker Hill & Sullivan Min. Co. April, 1910 to Oct., 1911, Mine Supt., Portland Canal Min. Co. Oct., 1911 to Jan. 1912, Sampling Federal Min. Co. Jan. to Nov., 1912, Engineering Office, W. M. Snow, Partner. Nov., 1912 to June, 1913, Mine Supt., Success Min. Co. June, 1913 to Jan., 1915, Genl. Supt., Consolidated Interstate Callahan Min. Co. Sept., 1915 to date, Genl. Supt., Rex Consolidated Min. Co.

Present position: Genl. Supt., Rex Consolidated Min. Co.

**Maxwell Naylor Short, Superior, Ariz.**

Proposed by I. A. Ettlinger, W. C. Browning, Henry C. Wilmot.

Born 1889, Pembina, N. Dak. 1906, Grad., Los Gatos High School, Los Gatos, Cal. 1911, B. S., College of Mining, Univ. of California. 1911-12, Shift-boss, Crushing Plant, El Rayo Min. Co., Santa Barbara, Chih., Mex. 1912-13, Engr., Draftsman and Sampler, American Smelters Securities Co., Santa Barbara, Chih., Mex. 1914-15, Sampler, Miner and Timberman, Detroit Copper Co., Morenci, Ariz. 1915-16, Miner and Timberman, Calumet & Arizona Min. Co., Bisbee, Ariz.

Present position: Asst. Engr., Magma Copper Co.

**Herbert C. Shotwell, Prescott, Ariz.**

Proposed by E. S. Smith, J. N. D. Gray, D. H. Bradley, Jr.

Born 1882, Bucyrus, Ohio. 1905, B. S. & E. M., Michigan College of Mines. 1905, Mining Engineer with Walter H. Wiley, Idaho Springs, Colo. 1906, Engr., Haggot-Girard Co., Prescott, Ariz. 1909-15, County Surveyor, Tavapai Co., Ariz. 1909-15, City Engr., Prescott, Ariz. 1913-14, Engr., United Verde Extension Mining Co., Jerome, Ariz.

Present position—1906 to date: Consulting Engineer.

**Homer Aldrich Sitterley, Low Moor, Va.**

Proposed by R. A. Lipscomb, F. U. Humbert, R. B. Ladoo.

Born 1889, Bath, Steuben Co., N. Y. June, 1909, Graduated from Haverling High School, Bath, N. Y. April, 1907 to April, 1913, Civil Engineering course with International Correspondence School. (Graduate, 1913.) Have since studied the Mining Engineering Course but without taking examinations. Sept. 1912-13, Civil Engineering in private practice, also as Asst. City Engr. for City of Howell, N. Y. for three years, under D. A. Proctor, C. E., also F. H. Robinson as City Engineer. March, 1916, Chief Engr., The Low Moor Iron Co. of Va.

Present position: Chief Engr., Low Moor Iron Co. of Va.

**Allister Knox Stewart, Mine La Motte, Mo.**

Proposed by J. F. Thompson, A. J. Meier, F. E. Butcher.

Born 1863, St. Louis, Mo. Up to 1874, Public and private schools of Sparta, Ill. and Washington, Mo. Franklin School, 1870-74. 1874-93, Mathematics under Prof. Blawett privately under Prof. Jameson and Prof. Morgan of St. Louis High School. Later under direction of Dr. H. T. Pritchett of Washington University. 1891-93, Literary work, *St. Louis Spectator* and other periodicals. Member of the Academy of Science in the early 90's. 1893-1917, Student of Science throughout my life, especially mining geology. Writer and contributor to literary and scientific publications on Natural Hist., Archeology, Geology, Mining, etc. 1883, Sorted ore for mine at Lawson, Colo. 1885, Employed by the Woody Creek Mining Co., Aspen, Colo. 1886, Ragged Mt. Placer Min. Co. on North fork of the Gunnison River, Colo. 1887, Florence C. Min. Co., Ouray, Colo. 1887-97, Interested in various min. enterprises throughout Mo., travelled 1,000 miles on mule back throughout the min. district of Honduras for St. Louis Syndicate. 1903, Installed and operated a placer plant on Grand River for St. Louis Syndicate. 1901, Examined and reported on 86 zinc and lead mines for the Frisco Ry. in Northern Ark. 1904, Engaged by E. H. Gorse, for the Drummond Estate to straighten out a rather complicated min. and met. situation with mines and mills at Montezuma, Colo. and smelter at Upland, Ind. Made exam. on group of 14 mines in Oaxaca and afterward accepted the management of the properties for the Mexico Gold & Silver Min. Co. 1906-11, Director and Genl. Mgr., St. Louis Oaxaca Iron & Steel Co. Consulting Engr., for G. J. Kobush. 1907-12, Interested with Benj. Nicol and Sir Weetman Pearson (now Lord Condroy) in mining in Mexico. Also with Ladenburg Thalman & Co., N. Y. 1913, Various examinations throughout Missouri and Arkansas. Examined New Cornelia Copper Co. Mines, Ajo, Ariz. 1912, Examined for St. Louis Syndicate iron mines of Missaba and Vermillion Ranges. 1912, Consulting Engr. for Maurice Peugeot on drilling operations at Bonne Terre. 1911, Appointed U. S. Rep. of Association Financiera International of Mexico. 1912, Third trip to mine districts of California for Jno. R. Boddie, St. Louis, Mo. 1913, Mined Zinc at Wentworth near Joplin and reported on four mines in Missouri and Arkansas. 1914, Contracted with Finley and Paige of Chicago to manage California mine for them. 1914, Made report for the Countess de Gorouche on California mine at Angels Camp. 1915, Employed by the Chicago and New York Syndicate to report on Mining possibilities of the Cobalt and Porcupine Districts (6 months). *Mining World* published my article on this district, April 15, 1916.

1916, Engaged by Western Development Co. who operate the Down town mines of Leadville, Colo. on special work.

Present position: Geologist, Mine La Motte, Mo. Metals Corpn.

**George Eustace Stott, Singapore, India.**

Proposed by F. L. Cole, G. E. Brown, A. L. Blomfield.

Born 1884, Oldham, Lancashire, Eng. 1897-1903, General Education, Hulme Grammar School, Manchester, Eng. 1905-08, Redrutt School of Mines, Cornwall, Eng. 1904, Apprenticed pupil, Hookersall & Co., Ltd., Manchester, Eng. 1908, Practical experience underground at Bassett Mine, Cornwall, Eng. 1909-10, Surveyor and Assayer, Darded Gold Mines, Panama. 1911-12, Reduction Works Asst., Ketahoen Gold Mines, Sumatra. 1912-14, Lorgkad Oil Co., Sumatra as Asst. Geol. 1915, Asst. Min. Engr., Lorgkad Oil Co., Sumatra.

Present position: Asst. Mining Engr., Lorgkad Oil Co.

**Karl Sulberg, Checotah, Okla.**

Proposed by A. R. Campbell, F. Y. Robertson, S. J. Jennings, C. F. Moore.

Born 1878, Tserlohn, Germany. 1884-88, Primary school, Tserlohn, Germany. 1888-92, High school, Tserlohn, Germany. 1892-95, Cie. des métaux and produits chimiques at Averbelt, Belgium, smelter. 1895-97, Technical College, Haasen. 1900-02, Univ. at Liege. 1897-1900, Cie. des métaux and produits chimiques, roasting, sulphuric acid making, spelter, lead, silver, arsenic, blue vitriol, etc., etc, charge of zinc smelting, Belgium. 1902-04, United States Zinc Co., Pueblo, Colo., metallurgist, building and starting zinc smelter. 1904-07, Sté. metalurgique de Lommel, Lommel, Belgium, building and operating zinc smelter, sulphuric acid plant, ingénieur en chef. 1907-10, Metalwerke Unterweser A. G. Nordenham, Germany, building and operating zinc smelter, sulphuric acid plant, manager of operations. 1910-12, Instruction trip, Europe, North Africa, office at Brussels, Belgium. 1913-15, American Zinc Co., Hillsboro, Ill., metallurgical engineer.

Present position: U. S. Smelting Co., Checotah, Okla., metallurgical engineer.

**Grover Cleveland Taylor, Republic, Wash.**

Proposed by S. H. Richardson, J. C. Haas, L. K. Armstrong.

Born 1884, Lake Valley, N. Mex. 1910-11, Special Work in School of Mines at W. S. C., Pullman, Wash.; Subject, Mining Engineering. Continuous employment in mining and met. work since 1900. 1910, Supt., White Ribbon Mine, Atlanta, Ida. 1911, Surveyor B. C. Copper Co., Greenwood, B. C. 1912-14, Supt., Sun Pail Mill, Republic, Wash. 1914-15, Assay & Eng. Office, Republic, Wash. 1916, In charge of Lease, Reid Mining Co., Whitehouse, Cal. 1916, Analyst, Bully Hill Mining Co., Winthrop, Cal. 1916-17, Lone Pine Surprise Con. Min. Co., Republic, Wash.

Present position: Mine Supt.

**Charles Lynn Thompson, Casper, Wyo.**

Proposed by A. C. Boyle, Jr., W. D. Waltman, C. T. Lupton.

Born 1892, South Bend, Wash. 1908, Graduated Wazata, Minnesota High School. 1908-09, Post Graduate work, School of Agriculture, Minn. 1909-10, Field work in engineering and teaching country school winter term at Cloverly, Wyo. 1910-11, Freshman year, Colorado School of Mines. 1914, Graduated University Wyoming, Bachelor of Science in Mining Engineering. Summer 1911, Municipal Water System construction with Katz-Craig Construction Co., Basin, Wyo. Summer 1912, Timberman, underground, for Colorado Fuel & Iron Co., Sunrise, Wyo. Summer 1913, Asst. Engr., Colorado Fuel & Iron Co., Sunrise, Wyo. Geological mapping and General Engineering. Summer 1914, powder monkey, rope rider, track man and helper on coal cutting machine, Union Pacific Coal Co., Superior, Wyo.

Present position: Consulting Engineer and Oil Geologist.

**Douglas Tuck, Pílares de Nacozari, Son., Mex.**

Proposed by McHenry Mosier, H. H. Retter, H. T. Hamilton.

Born 1872, London, England. 1880, General Education at Colfe Grammar School, London, and The Vale Academy, Ramsgate, Kent, Eng. 1892-95, Technical Education as Asst. to Sir Alec Rose Stenning, London, E. C., Eng. 1908, Obtained the degree of Associate Member Inst. C. E. by exam. also associate member South African Assn. of Engrs. 1896-98, Asst. Engr., Cons. of Bahía Blanca & N. W. R. R., Argentine Rep. 1899-1900, Engr., Portobelo Gold Mines, Zaruma, Ecuador, belonging to the So. American Dev. Co., New York. 1903-06, Engr. for Road Construc-

tion and Maintenance, Johannesburg Municipality, So. Africa. 1906-11, Min. down the Ferreira Deep Gold Mine, Johannesburg. 1911-15, Engr., Santa Gertrudis Mine, Pachuca, Mex. District Engr., Compama Real del Monte, Chief Engr., Compañia Minera de Penoles, near Mapimi, Durango, Mex. 1915-17, Mining in Junction Mine, Bisbee, Ariz. Practising as a Consulting Engr., Oatman, Ariz.

Present position: Geologist, Mochesuma Copper Co.

S. Power Warren, Baxter Springs, Kansas.

Proposed by W. A. Butchart, H. C. Parmelee, R. A. Leahy.

Born 1889, Denver, Colo. 1912, Colorado School of Mines. Degree of E. M.

Present position: Mgr., Rialto Mining Co., Carden, Okla.

Herbert Wirshing, Tulsa, Okla.

Proposed by D. M. Armstead, C. J. Adami, W. E. McCourt.

Born 1886, Sterling, Kansas. 1906, Public Schools, Sterling, Kansas. 1907, Cia. Carbonifera Agujita, S. A. 1908, Sabinas, Coah., Mex. 1909-10, San Antonio Machinery & Supply Co., San Antonio, Texas. 1911-13, Dodson Mfg. Co., Sen C., Torreon, Coah., Mex. 1914 to date, Ingersoll Rand Drill Co., St. Louis, Mo.

Present position: Salesman with Ingersoll Rand Co.

Richard Hoe Worcester, St. Louis, Mo.

Proposed by L. D. Ricketts, H. A. Fitch, T. H. O'Brien.

Born, 1881, West Campton, New Hampshire. Grammar and High School, Detroit, Mich. 1902, B. S., E. M., Michigan College of Mines. 1902-04, Mine Surveying, Oliver Iron Mining Co., Ironwood, Mich. 1904-06, Drafting, Surveying and Mill Work, Old Dominion Copper Mining & Smelt. Co., Globe, Ariz. 1906-08, Draftsman at Washery, Dawson Fuel Co., Dawson, N. Mex. 1908-09, Millman and Supply man, Concentrator, Steptoe Valley Smelting & Min. Co., McGill, Nev. 1909-11, Draftsman, Foreman Crushing Plant, Concentrator, Cananea Cons. Copper Co., Cananea, Mexico. 1911-12, Const. Engr., Stag Canon Fuel Co., Dawson, N. Mex. Farming. 1912-13, Mine Surveyor, United Verde Cons. Copper Co., Jerome, Ariz. 1913-17, Const. Engr., Stag Canon Fuel Co., Dawson, N. Mex.

Present position: Supt. Min. Dept. Laclede Christy Clay Products Co.

### *Associate Members*

Benedict J. Baker, Boston, Mass.

Proposed by J. A. Caselton, H. Garlicks, W. E. McCourt.

Born 1884, Springfield, Ky. High School. Two years private teacher. Ending June 1904, One year Japan Commercial College. 1904, Saleman. 1906, Buyer. 1909, Resigned to sell beet sugar companies bonds for R. P. Davis, Colo. Springs and Los Angeles. 1911, Resigned and started selling Investment Securities in Boston under the name of B. J. Baker & Co. 1912, Organized B. J. Baker & Co., Inc., Investment Securities (Mass. Laws) Boston, Mass. Have mining interest, Leadville, Colo., South East Missouri and Oklahoma.

Present position: Pres., B. J. Baker & Co., Inc., Boston, Mass.; Pres., Baker Lead Co., S. E. Mo.; Pres., Boston-Elvins Lead Co., S. E., Mo.

Robert E. Caskey, Pittsburg, Kansas.

Proposed by L. A. Delano, R. S. Foster, H. Rabling.

Born 1884, Philadelphia, Pa. General School and High School from 1890 to 1902 at Philadelphia, Pa. Received A. B. Degree, Princeton Univ., Class 1906. 1906-07, With Pa. R. R. in Maintenance of Way Dept. 1901-13, With E. I. Du Pont de Nemours Powder Co., in Technical Division.

Present position—1913 to date: Sales Agent, Atlas Powder Co.

John Parke Hood, Tyrone, N. Mexico.

Proposed by J. Archibald, H. Dolan, E. C. Luther.

Born 1893, Philadelphia, Pa. 1907-08, Mercersburg Academy. No degrees. 1911-16, Susquehanna Coal Co., Engineer Corps, Lytle Colliery. 1916-17, Service on Mexican Border as Serg. 1st Class, Co. C, Penna. Engineers.

Present position: Sampling, Phelps-Dodge Corp., Tyrone Branch.

William E. Merritt, Nashville, Tenn.

Proposed by G. P. Hulst, Fred P. Clark, R. Ruetschi.

Born, Springfield, Ill., 1881. 1905, Grad., U. S. Military Academy. Served as 2d Lieut., U. S. Army in Philippines to 1908. Resigned to enter business.

Present position—1908 to date: Supt., Armour Fertilizer Works and Tennessee Chem. Co.

**Charles E. Pope, Pittsburgh, Pa.**

Proposed by Henry C. Carr, James Gayley, John A. Topping.

Born 1859, Pittsburgh, Pa. Mfg. Steel, Tin Plate, Fire Brick, etc., etc. President of the following companies: Alpha Portland Cement Co., Pope Tin Plate Co., Standard Seamless Tube Co., Impervious Metal Co., Chairman and Director, American Metal Cap Co., Vice-President, Proprietary Mines Co. of America.

Present position: Steel Mfg.

### *Junior Members*

**E. J. Benoist, Cleveland, Ohio.**

Proposed by F. R. Van Horn, J. Burns Read, Z. Jeffries.

Born 1893, Santiago, Chile. 1912, Instituto Nacional, Santiago, Chile. 1917, Will graduate from mining dept., Case School of Applied Science.

Present position: Student, Case School of Applied Science.

**James Marion Bugbee, Boston, Mass.**

Proposed by C. E. Locke, E. E. Bugbee, H. O. Hofman.

Born 1896, Boston, Mass. Sept. 1914 to present, Student in Metallurgy at Mass. Institute of Technology.

Present position: Student at Massachusetts Institute of Technology.

**Ezra Edward Erich, San José, Cal.**

Proposed by H. W. Young, C. F. Tolman, Jr., G. H. Clevenger.

Born 1894, Tacoma, Wash. 1910-14, San José High School, San José, Cal. 1914-17, Stanford University, 1914. Prospecting work in Oregon Property owned by W. J. Erich. Summer 1915, Engineering for R. Flemming, San José, Cal. Summer 1916, Copper Queen Cons. Min. Co., Bisbee, Ariz.

Present position: Student, Mining Engineering at Stanford University.

**Artileus V. Eulich, Rolla, Mo.**

Proposed by C. R. Forbes, Chas. Y. Clayton, H. T. Mann.

Born, 1897, St. Joseph, Mo. 1911-15, St. Joseph Central High School. 1915-17, Missouri School of Mines. 1916, Summer, Inspiration Cons. Copper Co.'s Mine.

Present position: Student, Missouri School of Mines.

**Oscar Gotsch, Jr., Rolla, Mo.**

Proposed by Chas. Y. Clayton, A. L. McRae, H. A. Buehler.

Born, 1890, St. Louis, Mo. 1905, Garfield School, St. Louis, Mo. 1910, McKinley High School. Missouri School of Mines.

Present position: Student, Missouri School of Mines.

**Harlan C. Harbicht, Champaign, Ill.**

Proposed by H. H. Stoeck, E. A. Holbrook, C. M. Young.

Born 1896, Hannibal, Mo. 1910-14, Hannibal High School. 1914-15, Missouri School of Mines. 1915 to date, University of Illinois. 1915-16, Summer Work, Engineering Dept., The Atlas Portland Cement Co., Hannibal, Mo.

Present position: Student.

**Orie N. Maness, Rolla, Mo.**

Proposed by Chas. Y. Clayton, A. L. McRae, H. A. Buehler.

Born 1895, Webb City, Mo. 1914, Webb City High School, Webb City, Mo. 1914-17, Missouri School of Mines. 3 months, Pitcher Lead Co., Pitcher, Okla. 2 years, Field Mining & Milling Co., Benton, Wis. 4 months, Carmean & Squires, Webb City, Mo.

Present position: Student, Missouri School of Mines.

**Robert Marston, Rolla, Mo.**

Proposed by H. T. Mann, C. R. Forbes, C. Y. Clayton.

Born 1895, Chicago, Ill. 1911-14, El Paso High School. 1915-17, Missouri School of Mines, Rolla, Mo. 1912, Arizona Copper Co. 1914, Arizona Copper Co. 1915, Inspiration Copper Co.

Present position: Student, Missouri School of Mines.

**Paul F. Pape, Rolla, Mo.**

Proposed by A. L. McRae, C. R. Forbes, H. T. Mann.

Born 1896. 1909, Graduated Cape Girardeau, Mo., Grammar School. 1913, Graduated Cape Girardeau, Mo., High School. 1913-17, Missouri School of Mines,



**Rolla, Mo.** Summer sessions, 1910-13, Construction work, Pape Bros., Construction Engineers, Cape Girardeau, Mo. Summer session of 1916, Construction work, Doe River Lead Co., Mill No. 3, Elvins, Mo.

Present position: Student, Missouri School of Mines, Rolla, Mo.

**William Houston Reber, Rolla, Mo.**

Proposed by Chas. Y. Clayton, A. L. McRae, H. A. Buehler.

Born 1896, Jackson, Miss. 1910-14, Grad., Dallas High School. Missouri School of Mines, 3 years. Summer work: 1915, Draftsman; 1916, Instrument man.

Present position: Student, Missouri School of Mines.

**Leroy Robert Scheurer, Rolla, Mo.**

Proposed by C. R. Forbes, A. L. McRae, H. A. Buehler.

Born 1893, Washington Co., Ill. 1912, Graduated from Wichita Falls, Texas High School. 1917, Senior, Missouri School of Mines, Rolla, Mo. 1912, Wichita Motor Co., Wichita Falls, Tex. 1913, Buffalo Mines, Ltd., Cobalt, Ont. 1915, Pinos Altos Min. & Milling Co., Pinos Altos., N. Mex. 1916, Chino Copper Co., Hurley, N. Mex.

Present position: Senior Student, Missouri School of Mines.

**Leslie C. Skeen, Rolla, Mo.**

Proposed by C. R. Forbes, D. C. Kemp, H. A. Buehler.

Born 1893, Big Stone Gap, Va. 1911-12, Univ. of Va. Sept., 1914-Feb., 1915, Colorado School of Mines. Feb., 1915 to date, Missouri School of Mines. Dec., 1913-Aug., 1914, Transitman. General Office Man for Fox & Peck, Civil & Mining Engrs. of Big Stone Gap, Va. 1912-13, Clerical Work for C. S. Carter & Co., Big Stone Gap, Va.

Present position: Student, Missouri School of Mines.

**Harold Ewing Spickard, Stanford University, Cal.**

Proposed by G. H. Clevenger, H. W. Young, Welton J. Crook.

Born 1895, Spickard, Mo. 1913-15, Drake Univ. 1915-16, Mo. School of Mines. 1916-17, Stanford Univ. 1909-12, High School, Cameron, Mo. 1912-13, High School, West Des Moines, Ia. 1916, Summer, Copper Queen Cons. Min. Co., Bisbee, Ariz.

Present position: Student, Stanford University, Geology and Mining Dept.

**George Bongjo Tseo, Butte, Mont.**

Proposed by C. H. Clapp, C. H. Bowman, B. Gore.

Born 1893, Wanchang, China. 1910-13, Student at Evanston Academy, Northwestern University and graduate of the same. 1913-14, Student at Mich. College of Mines. 1914-16, Colo. School of Mines. 1916-17, Mont. State School of Mines.

Present position: Student, Mont. State School of Mines.

**Carol John Wakenhut, Lawrence, Kansas.**

Proposed by A. C. Terrill, C. L. Severy, W. E. McCourt.

Born 1895, Salina, Kansas. 1910, Salina Public Schools. 1910-14, Salina High School. Present time, Senior in University of Kansas. 1913, Chairman, Winnipeg, Salina & Gulf Ry. 1915, Joplin District. 1916, Instrumentman, Carter Oil Co., Tulsa, Okla.

Present position: Student, University of Kansas.

**Fred Melville Wolverton, Oshkosh, Wis.**

Proposed by C. K. Leith, W. J. Mead, R. S. McCaffery.

Born 1890, Oshkosh, Wis. To 1910, Public Schools, Oshkosh, Wis. 1910-17, Univ. of Wisconsin. 1912-15, State of Wisconsin, Wis. Highway Commission. Summer 1916, Geological Survey of Canada.

Present position: Student, University of Wisconsin.

#### CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Apr. 10, 1917 to May 10, 1917.

This list together with the list published in Bulletin Nos. 121 to 125, January to May, 1917, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1917 and brings it up to the date of May 10, 1917.

- AARONS, JULIAN B., Major, Australian Imperial Force,  
74 Upper Gloucester Pl., Dorset Square, London, W., England.
- ADDAMS, CHARLES E. .... Genl. Mgr., Arizona Hercules Copper Co., Ray, Ariz.
- ALLEN, CHESTER A. .... 405 Hancock St., Peoria, Ill.
- ALLEN, HEMAN H. .... Perseverance Mine, Juneau, Alaska.
- ALLEN, MILTON A. .... Federal Lead Co., Flat River, Mo.
- ANDERSON, CARL N., Supt., Josevig Kennecott Copper Co.,  
Shushanna Junction, McCarthy, Alaska.
- ANDERSON, JAMES W., Patent Attorney, Duell, Warfield & Duell,  
25 West 44th St., New York, N. Y.
- AMBLER, J. O. .... Clifton, Ariz.
- AYER, FRANK A. .... Tyrone, New Mexico.
- BARAGWANATH, JOHN G. .... Casilla 309, Lima, Peru, South America.
- BARNARD, CLARENCE W., Care North Chicago Hospital, 2551 North Clark St.,  
Chicago, Ill.
- BATCHELLER, JAMES H., Min. Engr., Mgr., Virginia Lead & Zinc Corp.,  
Valcooper Mine, Mineral, Va.
- BATES, BENNETT R. .... Baxter Springs, Kansas.
- BAUMGARTEN, KARL. .... Instructed to hold all mail.
- BERRIEN, CHAUNCEY L., Asst. Genl. Supt. of Mines, Anaconda Copper Min. Co.,  
Box 1375, Butte, Mont.
- BLANTON, EDWARD A., JR. .... Philadelphia Bourse, Philadelphia, Pa.
- BLOOD, GEORGE D., Min. Engr., Consulting Practice, Judge Min. & Smelt. Co.,  
1021 Kearns Bldg., Salt Lake City, Utah.
- BOSTWICK, WILLIAM A., Asst. to the Pres., The International Nickel Co.,  
43 Exchange Place, New York, N. Y.
- BOWLES, ROSS. .... Supt., American Zinc Co. of Ill., Box 347, East St. Louis, Ill.
- BOYDELL, HARRY C. .... 3 Gladstone Road, Watford, Herts, England.
- BOYER, SAMUEL L. .... General Delivery, San Francisco, Cal.
- BRINELL, J. A. .... Nassjo, Sweden.
- BRINGS, HENRI. .... 724 Leavenworth St., San Francisco, Cal.
- BRUNS, CHRISTOPHER L., JR. .... Majagua, Camaguey, Cuba.
- BRUNTON, FREDERIC K. .... Supt., Cons. Arizona Smelting Co., Humboldt, Ariz.
- BURNS, JAY J. .... 35 N. Main St., Butte, Mont.
- BUTLER, N. C. .... Delmar, Oregon.
- CAINE, MILTON A. .... Asst. Mgr., Tennessee Copper Co., Copperhill, Tenn.
- CAMPBELL, ARTHUR R. .... P. O. Box 1133, Kansas City, Mo.
- CAPLES, RUSSELL B. .... The Leland Apartments, Great Falls, Mont.
- CARNAHAN, JOHN S. .... Supt., Mexican Lead Co., Apartado 115, Monterrey, Mexico.
- CARSON, ELLARD W., Mgr., Oceanic Quicksilver Mine, Oceanic Quicksilver Co.,  
Cambria, San Luis Obispo Co., Cal.
- CARTER, EDWARD E. .... Mgr., Gold Hill Mine, Quartzburg, Idaho.
- CAVANAGH, JOHN L. .... Mitchill Bldg., Halifax, N. S., Canada.
- CHARLETON, A. G., 5 Avonmore Road, West Kensington, London W. 14, England.
- CHATIN, AUGUST H. .... 3832 Umatilla St., Denver, Colo.
- CLARK, A. W., JR. .... 2222 Scottwood Ave., Toledo, Ohio.
- CLARKE, ROY H., Min. Engr., Mgr., Queen of Bronze Mine, Takilma Smelt. Co.,  
Takilma, Josephine Co., Ore.
- CODY, BENJAMIN H. .... Ajo, Arizona.
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## NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Apr. 10 to May 10, 1917.

| Date of Election. | Name.                   | Date of Death. |
|-------------------|-------------------------|----------------|
| 1911              | Bailey, A. C.....       | Mar. 21, 1917. |
| 1871              | Eilers, Anton.....      | Apr. 22, 1917. |
| 1916              | Teddy, Frederick T..... | — —, 1917.     |
| 1875              | Witherbee, Frank S..... | Apr. 13, 1917. |

## BIOGRAPHICAL NOTICE OF DAVID H. BROWNE

David H. Browne was born June 8, 1864, at Hollymount, County Mayo, Ireland. At the age of 16, after going through the Londonderry Academy, he came to this country, and entered the University of Michigan, where he was graduated in 1885. He began his scientific practice as chemist, under F. A. Emmerton, in the laboratory of the Joliet Steel Co., after which he served some time as chemist at the Ludington mine, Iron Mountain, Mich. But in 1888 he went back for a year to his alma mater as instructor in inorganic analysis. Those who knew him feel sure that he would have been an efficient and popular professor; but his ardent temperament could not brook the monotony and confinement of the classroom; and in 1899 he went to work at the Andrews and Hitchcock blast-furnaces, at Hubbard, Ohio, where he remained until 1891, when he entered as chemist the service of the newly organized Canadian Copper Co., operating in the Sudbury district, Canada. This was the beginning of his special professional life-work. For twenty-six years he devoted himself to the numerous and complex problems successively arising in connection with the metallurgical treatment of the nickel-copper ores of Sudbury.

One of his notable achievements was the development of an electrolytic process for the separation of nickel and copper in the ores referred to. This method was successfully employed for several years; but when the Canadian Copper Co. was absorbed by the International Nickel Co., the cheaper Orford process owned by the latter was substituted.

A still more permanently important piece of metallurgical progress credited to Mr. Browne was the successful firing of reverberatory smelting-furnaces with powdered coal—a measure which had been desired, dreamed and attempted by others, but which he was acknowledged as first to put in practice on a large scale.

The *Engineering and Mining Journal* of April 7, 1917, says editorially of this improvement:

"The coal-dust firing of reverberatory smelting-furnaces is something that in metallurgic and commercial importance ranks in the metallurgy of copper with the introduction of electrolytic refining and of basic converting. It was one of those fundamental innovations that reduced the cost of smelting, not by a little nibble, like the minor improvements, but by a great chunk. Although Browne was not the man in whose brain this conception originated, he was distinctly the man who first did the thing successfully, and made it a part of the art of smelting where other good men had failed. Browne's achievement in this was acclaimed by the entire profession of metallurgists without dissent. And it was characteristic of the man that, having done this great thing, he freely and generously communicated his knowledge to others, so that they might do it too."



DAVID H. BROWNE.

The commercial importance of this improvement may be measured by the surprising economy of fuel at the Anaconda, Montana, where seven tons of ore were smelted by its use with the amount of coal that had previously been required for four tons.

While Mr. Browne's eminence as a metallurgist became universally known through his work at Sudbury, the charm of his personal character and accomplishments was felt by comparatively few, so long as he re-

mained in the seclusion of that district. But after he was transferred to the New York Office of his company, in 1914, he became speedily and sympathetically related to many professional colleagues. He had joined this Institute in 1900, and the Canadian Mining Institute in 1905. As the chairman of our New York section, he soon became a leader in its social as well as technical activities. The last months of his life were devoted to the work of his friend Hoover, for the relief especially of the "Belgian Kiddies."

No longer ago than the last week of last February, Mr. Browne presided, full of energy and humor, at the informal reunion of members of this Institute, in connection with its Annual Meeting. On that occasion, he read a sketch of his own, a reminiscence of his Irish childhood, the delicate, quaint humor of which its hearers will not soon forget. It was a terrible shock to us all—his sudden death so soon thereafter. But now, as we look back upon it, we are grateful that our latest memory of him presents to us a picture of the genial, inspiring, generous comrade as well as the honored technical leader, the graceful and forcible speaker and the accomplished author.

Mr. Browne married, in 1889, Miss Sara M. Davis, who survives him, together with three sons. He died March 30, 1917, after a brief illness, at his home in Montclair, N. J.—in his 53d year—a sudden eclipse at noon of a shining life. Yet an eclipse is but a passing shadow; and, when the shadow flees away, behold! the sun, which has been shining all the while!



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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Ferrous Iron Content and Magnetic Susceptibility of Some Artificial and Natural Oxides of Iron

R. B. SOSMAN\* AND J. C. HOSTETTER,\* WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

### INTRODUCTION

It is well known that ferric oxide,  $\text{Fe}_2\text{O}_3$ , is paramagnetic, while magnetite,  $\text{Fe}_3\text{O}_4$ , is classed among the highly ferromagnetic substances. But magnetic data on oxides intermediate in composition between  $\text{Fe}_2\text{O}_3$  and  $\text{Fe}_3\text{O}_4$  have been almost completely lacking.

$\text{Fe}_2\text{O}_3$  and  $\text{Fe}_3\text{O}_4$  form a solid-solution series, according to present evidence.<sup>1</sup> In this series the properties change continuously from  $\text{Fe}_2\text{O}_3$  toward  $\text{Fe}_3\text{O}_4$ , as the percentage of FeO increases from zero toward 31.03, which is the percentage in magnetite. It is possible that there is a break in the series near  $\text{Fe}_3\text{O}_4$ , but it has not yet been possible to establish the existence of such a break experimentally.

The occurrence of solid solution in this series is shown by the dissociation pressure, or oxygen pressure in equilibrium with the oxides, which falls continuously over the range from  $\text{Fe}_2\text{O}_3$  to  $\text{Fe}_3\text{O}_4$ . The oxygen-pressure curves at 1,100° and 1,200° are shown in Fig. 1. The existence of solid solution is also demonstrated by the continuous change in optical properties from  $\text{Fe}_2\text{O}_3$  over to a composition containing about 18 per cent. FeO, at which point the opacity of the oxide becomes so great that it is impossible to obtain measurements at higher percentages of ferrous iron.

Natural oxides of iron intermediate in composition between  $\text{Fe}_2\text{O}_3$  and  $\text{Fe}_3\text{O}_4$  are much more common than is generally supposed. If a mineral oxide is not strongly attracted by a small hand magnet and if it gives a red streak, it is usually labeled "hematite." If it is strongly attracted by the hand magnet it is usually labeled "magnetite" without further tests. As we shall show later, oxides containing from 1 up to 31 per cent. FeO can be thus erroneously lumped together as "magnetite." It is a fact, nevertheless, that the great bulk of natural oxides of iron lie fairly near either  $\text{Fe}_2\text{O}_3$  or  $\text{Fe}_3\text{O}_4$  in composition. The reason will be

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<sup>1</sup> R. B. Sosman and J. C. Hostetter: The Oxides of Iron, *Journal of the American Chemical Society* (1916), 38, 807-833.

evident from the shape of the dissociation-pressure curves of Fig. 1. If drawn to a direct scale instead of a logarithmic scale of pressures, the middle portion of the curves would be decidedly flat, with a rapid rise near  $\text{Fe}_2\text{O}_3$  and a rapid fall near  $\text{Fe}_3\text{O}_4$ . It is clear, therefore, that at a given temperature only a certain narrow range of oxygen pressures is available for the production of oxides of say 2 to 29 per cent.  $\text{FeO}$ , while all other pressures would produce oxides approaching pure  $\text{Fe}_2\text{O}_3$  or pure

*Minerals:*

*M* = Magnetite

*H* = Hematites

*Marlites* (pseudomorphs)

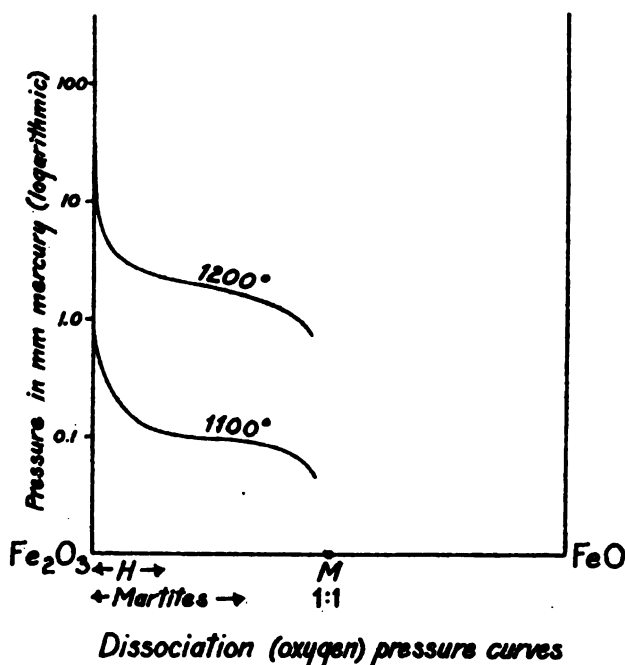


FIG. 1.—OXYGEN PRESSURE CURVES IN THE SYSTEM  $\text{Fe}_2\text{O}_3$ - $\text{Fe}_3\text{O}_4$ .

$\text{Fe}_3\text{O}_4$  in composition. The solid-solution series of the oxides of iron thus differs from the solid-solution series of the lime-soda feldspars in that the probability of occurrence of any given member of the feldspar series is much more independent of the composition than is the probability of occurrence of any given member of the iron oxide series.

Most of the hitherto published analyses of iron ores and iron oxide minerals show the total iron, but not the amount of iron which is in the ferrous condition. We find that practically all natural oxides of iron contain a determinable amount of  $\text{FeO}$ . Specular and micaceous hema-

tites, especially, show notable percentages of ferrous iron, and at the same time are often markedly magnetic. The experiments reported on in this paper were undertaken in order to find what relation exists between the ferrous iron percentage and the magnetic properties.

That there is some relation between the two was recognized as long ago as 1848 by Plücker,<sup>2</sup> who observed that some hematites were much more magnetic than others. He was of the opinion that the magnetism is due to small percentages of FeO, which he thought might be determined quantitatively by measuring the susceptibility of the oxides.

The term "magnetic" as applied to a mineral usually means little more than that the mineral is noticeably attracted by a small horseshoe or bar magnet. The impression gained from such tests is that the magnetic susceptibility increases much more rapidly than in proportion to the percentage of FeO, since a hematite with only a fraction of 1 per cent. FeO can be picked up by a hand magnet just as magnetite itself can be picked up. But the field of such a magnet is so irregular, and so intense close to the iron, that it gives no adequate quantitative idea. Measurements presented in this paper show, in fact, that the magnetic susceptibility seldom exceeds a value proportional to the percentage of FeO.

#### APPARATUS AND METHOD

The comparative magnetic susceptibility of the natural and artificial oxides of iron was determined by weighing the force exerted upon a known weight of the substance by a non-uniform magnetic field.

The apparatus used for the magnetic tests, as shown in Fig. 2, consisted simply of a Rueprecht analytical balance, from one pan of which was suspended a glass platform on which the charge of oxide was placed, a short distance above the poles of an electromagnet. The oxide was contained in a small flat-bottomed glass pan, and the particles were kept from moving by a glass plunger which pressed down the powder. The steel

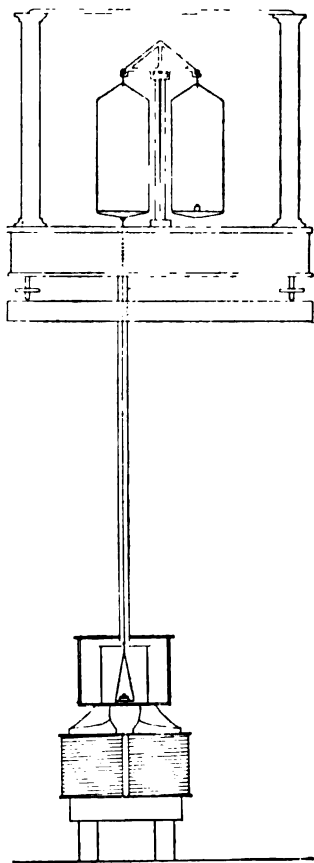


FIG. 2.—APPARATUS FOR COMPARATIVE MAGNETIC TESTS.

<sup>2</sup> Plücker: Über Intensitätsbestimmung der magnetischen und diamagnetischen Kräfte. *Poggendorff's Annalen* (1848), 74, 321-379.

pointer of the balance was replaced by a pointer made of brass; the polarization of the steel pointer would otherwise have caused an appreciable deflection, necessitating a large correction to the results. The steel knife edges caused only a small deflection, easily corrected for. Blank measurements made with no oxide in the glass pan showed a maximum correction of 0.11 mg. One gram of crystallized powdered ferrous ammonium sulphate, at the maximum magnetizing current, was attracted by a force of 2.68 mg., which could be determined within 0.03 mg.

The distance of the glass platform above the poles was 5.1 mm. A variation of 1 mm. in this distance was found to produce a change of 0.22 mg. in the attraction, in a total of 6.02 mg. The error due to movement of the balance or electromagnet was therefore inappreciable, as such movements were found to be of the order of 0.1 mm.

This simple form of apparatus serves very well for comparative tests, and it is possible to obtain with it data on the true susceptibility of paramagnetic substances, in which the susceptibility does not vary appreciably with the field strength. But the field is rather concentrated, and it is difficult to determine what is the value of the magnetizing force at a given point. The iron core of the electromagnet introduces the further complications that the magnetizing force is not only a complex function of the current, but also is different according as the current is being increased or decreased, on account of the hysteresis of the core. As we wished to obtain comparative data not only on susceptibility but also on the relation between magnetizing force and induction, as well as on the remanence and coercive force, of the entire series of oxides to be examined, the electromagnet was later replaced by a set of solenoids of known constants, and other improvements were made in the apparatus at the same time. The new solenoid apparatus is now being used for further experiments, but the data given in this paper were all obtained with the simpler electromagnet form.

### *Analysis of Samples*

The ferrous iron was determined by solution in hydrochloric acid in an atmosphere of carbon dioxide, and titration with potassium permanganate. The method has been described in detail in a previous publication.<sup>3</sup>

Percentages of FeO of the order of 0.1 per cent. or less cannot be very accurately determined by titration with permanganate in the presence of chlorides, even with the addition of the usual "titrating mixture," as the quantity of permanganate solution required is not much more than that necessary to give the end-point color. Samples containing very little FeO were therefore titrated electrometrically, by measurement of the oxidation-reduction potential of the solution, using a platinum elec-

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<sup>3</sup> J. C. Hostetter: *Journal of the American Chemical Society* (1916), **38**, 821.

trode against a tenth-normal calomel electrode, with dilute potassium dichromate as the titrating solution. A few thousandths of a per cent. of ferrous iron can be detected and estimated by this means. Description and tests of the method will be presented in a later publication.

As will be evident from the results presented later in Tables 5 and 6, the permanganate method, though it gives only an approximate value when the percentage of FeO is small, agrees very well with the electrometric dichromate method in the cases where both were tried on the same sample. The difference is large only in the case of No. 1,521, on which electrometric gave 0.275 per cent., per manganate 0.15 per cent. The error in the permanganate titration in this case must have been accidental. The very small quantity of sample available did not permit of a repetition of the analysis.

### *The Ferromagnetic Members of the Series*

For paramagnetic substances, such as ferrous ammonium sulphate and ferric oxide, the weight and shape of the charge and its distribution in the magnetic field have very little effect upon the magnetic attraction. But as the susceptibility rises in the series and the oxides become more ferromagnetic, the attraction is affected by several variables besides the amount of ferrous iron in the sample. It is a characteristic of the so-called ferromagnetic substances that the magnetization in a given field varies with the shape of the fragment, with the fineness of grain (if the substance be powdered), with the distribution of the material in the magnetic field, and with the nearness of the particles to each other. In addition, the particles often possess a certain remanence, that is, become permanently polarized, so that their attraction in a given field also depends upon their previous magnetic history. The weighing of the magnetic attraction, therefore, seems to the physicist a very crude means of getting at the magnetic properties of a ferromagnetic material, properties which are perfectly definite and measurable by familiar methods, provided only that a large and homogeneous sample can be obtained. But large size and homogeneity are just the characteristics which are most conspicuous by their absence in the case of natural minerals, and we must therefore be content with a method which permits the use of very small quantities of powdered materials.

By powdering to about the same degree of fineness and distributing the samples similarly, good comparative results can be obtained, as will be shown in the following section.

### TESTS ON MAGNETITE

For comparison with the results on natural and artificial hematites, and to test the reliability of the method, a series of measurements was

made on magnetite and magnetite-quartz mixtures. Since magnetite is the most ferromagnetic member of the  $\text{Fe}_2\text{O}_3$ - $\text{Fe}_3\text{O}_4$  series, it is to be expected that the variations in comparative results obtained by changing the variables will be at a maximum in the case of magnetite, and will be less for other members of the series in proportion to their smaller magnetic susceptibility.

Crystals of magnetite from the Barton Hill mine at Mineville, N. Y. (sample 3b), were crushed and sized through Tyler Standard sieves. The portion which passed 325 mesh (opening 0.038 mm.) was further fractionated by means of an air jet, in an air elutriator similar to that used by the Bureau of Standards for the classification of cement,<sup>4</sup> but of smaller size.

In the earlier tests, the magnetizing currents used were 110, 400, 800, and 400 milliamp., in the order named. In order to obtain a better indication of the shape of the magnetization and hysteresis curves of the specimens, later tests were made with currents of 515, 800, and 485 milliamp., since it was found experimentally that ferrous ammonium sulphate was attracted with just one-half the maximum force (due to 800 milliamp.) by a rising current of 515 or a falling current of 485 milliamp. For the present we are interested only in the comparative attractions produced by the maximum magnetizing current of 800 milliamp.

The reproducibility of the measurements was tested on 500-mg. portions of the magnetite powder between 170 and 200 mesh (sieve openings 0.089 to 0.074 mm.). The results are shown in Table 1.

TABLE 1.—*Reproducibility*

|   | Mg. per Gram |
|---|--------------|
| New sample.....                           | 9,330        |
| Same, taken out and thoroughly mixed..... | 9,258        |
| Same, taken out and thoroughly mixed..... | 9,261        |
| New sample.....                           | 9,282        |
| Mean.....                                 | 9,283        |

The reproducibility of the measurements is thus well within 1 per cent.

The effect of the weight of the charge upon the attraction per gram was tested on the same size powder (170 to 200 mesh) with the result shown in Table 2.

TABLE 2.—*Effect of Size of Charge*

| Mg.      | Mg. per Gram |
|----------|--------------|
| 500..... | 9,283        |
| 350..... | 9,395        |
| 100..... | 9,549        |

<sup>4</sup> J. C. Pearson and W. H. Sligh: An Air Analyzer for Determining the Fineness of Cement. U. S. Bureau of Standards, Technologic paper No. 48 (1915).

For a fivefold increase in the quantity of oxide tested, there is an increase of about 3 per cent. in the attraction per gram.

The size of the particles has a considerable effect, as shown by the figures in Table 3, obtained on 500-mg. charges.

TABLE 3.—*Effect of Size of Particles*

| Size               | Diam. Particles,<br>Mm. | Attraction<br>Mg. per Gram | Per Cent.<br>FeO | Per Cent. In-<br>soluble Residue |
|--------------------|-------------------------|----------------------------|------------------|----------------------------------|
| 100-115 mesh.....  | About 0.16              | 9,516                      | 29.66            | 2.62                             |
| 170-200 mesh.....  | About 0.12              | 9,283                      |                  |                                  |
| 300-325 mesh.....  | About 0.07              | 8,982                      |                  |                                  |
| 2-mm. air jet..... | 0.05 and smaller        | 8,718                      | 29.62            | 2.71                             |

Before the effect found can be ascribed to difference in size of the particles, it must first be shown that the composition of the finest material is not markedly different from that of the coarsest material. Not only might the finest powder be expected to be more oxidized than the coarse, but the insoluble quartzitic residue might be expected to be unequally distributed among the fractions. The results shown in the last two columns of Table 3 were obtained by solution in HCl under a carbon dioxide atmosphere followed by electrometric titration. The figures show that the coarsest and finest fractions are practically identical in composition. The difference in magnetic attraction, amounting to about 9 per cent., must therefore be ascribed to the difference in size of the particles.

Taken as a whole, the tests show that satisfactory comparative results can be obtained by the method of weighing the attraction in a strong field, even on a ferromagnetic oxide, and that if desired the effect of certain variables can be evaluated. But for comparison over a range of 1 to 5,000, which is approximately the range in the hematite-magnetite series, even variations such as those due to different sizes of particles are of secondary importance, and comparisons can be made without carefully sizing the powder, which might be difficult to do on such small quantities as were available from some of the samples.

TABLE 4.—*Effect of Dilution of Magnetite*

|  |       |       |     |      |
|--|-------|-------|-----|------|
| Percentage of original magnetite in mixture.....   | 100   | 50    | 10  | 1.0  |
| Magnetic attraction, mg. per gram of mixture:  |       |       |     |      |
| 1. If directly proportional to percentage.....   | 8,718 | 4,359 | 872 | 87.2 |
| 2. If proportional to percentage, but corrected for<br>varying amount of magnetite present (Table 2).... | 8,718 | 4,441 | 900 | 90.3 |
| 3. As found experimentally.....  | 8,718 | 3,896 | 735 | 69.7 |

The effect of dilution of the magnetite was tested by mixing weighed

portions of the 2-mm. jet size magnetite with powdered quartz which had been ground to pass 200 mesh (0.074 mm. and finer). The results are shown in Table 4.

These results show that dilution of the magnetite diminishes the induction by an amount which is greater, the greater the dilution. This is to be expected, because placing the individual particles

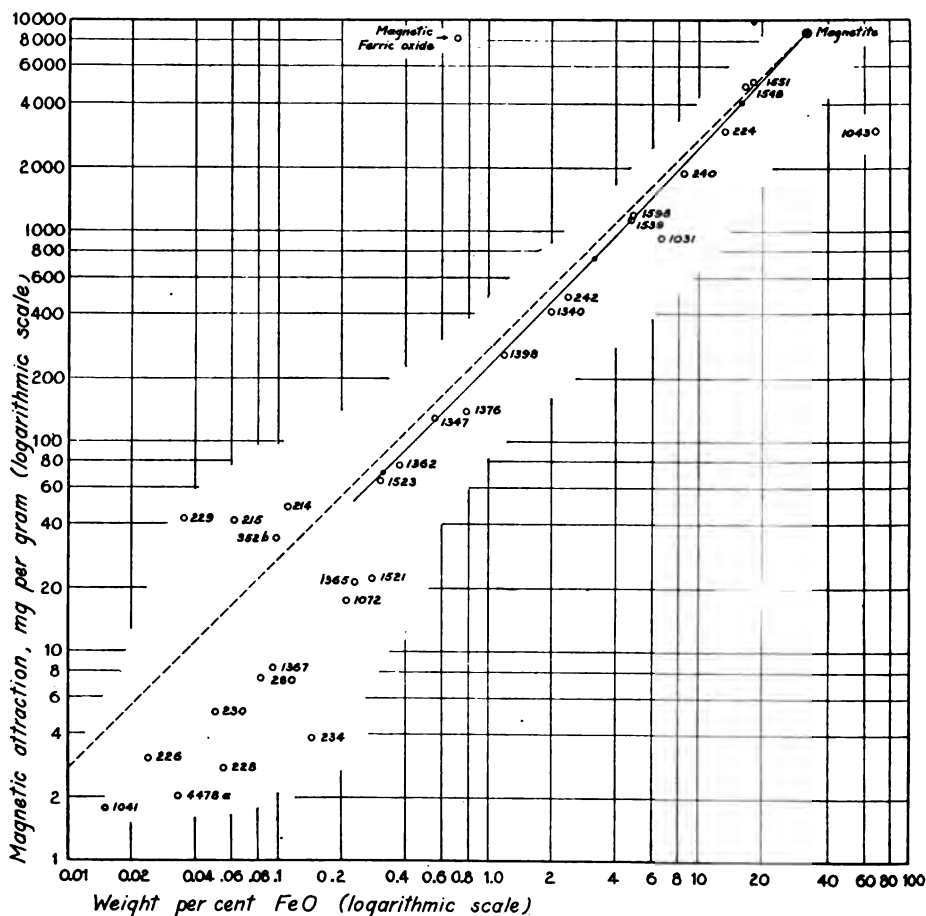


FIG. 3.—RELATION OF MAGNETIC ATTRACTION TO PERCENTAGE OF  $\text{FeO}$ , ARTIFICIAL OXIDES.

farther apart should diminish their mutual inductive effect. The results are shown graphically by the solid curve and the black dots in Fig. 3. The broken curve in the same figure represents the relation that would exist between magnetic attraction and percentage of ferrous iron if the attraction were strictly proportional to the amount of magnetite present.



The existing data on the dilution of magnetic powders are not very complete and are also not in very good agreement. Auerbach<sup>5</sup> used very finely divided chemically reduced iron powder, and diluted it with wood powder. The density of the iron in the mass was varied from 0.0176 to 3.612. Within this range the magnetization increased with the density, not linearly nor uniformly, but according to a curve which is expressible as a power series beginning with a term which expresses the magnetization at infinite dilution.

Benedicks<sup>6</sup> diluted magnetite (size between 0.4 and 0.2 mm. sieves) with powdered quartz and obtained a curve of form somewhat different from Auerbach's, but agreeing with Auerbach in that the magnetization increased with the concentration of the magnetite. Iron-copper and nickel-copper powdered mixtures gave similar curves. Benedicks believes that the peculiar form of Auerbach's curve resulted from some unrecognized variable.<sup>7</sup>

Our curve, being in terms of magnetic attraction per gram and concentration by weight, is not directly comparable with the results quoted above, which are in terms of intensity of magnetization and concentration by volume. Our curve agrees with those in showing that the effect of dilution is to diminish the magnetization. The effect is less than that indicated by Benedicks' curves. This may be due to the fact that our field was much more intense than the field of 143 gauss used by Benedicks; as Maurain<sup>8</sup> has shown, the magnetization tends to become more nearly independent of the concentration as the field becomes stronger.

### *Oxidation of Magnetite by Grinding*

The following results on a mixture of quartz and magnetite suggested that  $\text{Fe}_3\text{O}_4$  could be oxidized to paramagnetic  $\text{Fe}_2\text{O}_3$  by grinding. The original mixture contained 0.635 per cent.  $\text{Fe}_3\text{O}_4$ :

|                       | Magnetic Attraction<br>at 600 Milliampères,<br>Mg. per Gram |
|-----------------------|---|
| Original mixture..... | 26.68   |
| Ground 15 min.....    | 22.58   |
| Ground 15 min.....    | 20.45   |
| Ground 15 min.....    | 19.08   |

It was not certain from these tests whether the change was due to increased fineness, or whether oxidation was partly responsible. Analyses

<sup>5</sup> F. Auerbach: *Magnetische Untersuchungen. Wiedemann's Annalen* (1880), 11, 353-394.

<sup>6</sup> C. Benedicks: *Experiments on Allotropy of Iron: Behaviour of Ferromagnetic Mixtures; Dilatation of Pure Iron. Journal of the Iron and Steel Institute* (1914), 407-459.

<sup>7</sup> *Op. cit.*, 414.

<sup>8</sup> C. Maurain: *Sur les propriétés magnétiques des poudres de fer. L'Éclairage électrique* (1903), 34, 465-472.

TABLE 5.—*Ferrous Iron Content and Comparative Susceptibility of Artificial Oxides of Iron in Powder Form*

| Specimen No. | Magnetic Test No. | Designation of Specimen  | Source                            | Color of Powder |  | Wt. of Sample Tested, Mg. | Temp. | Per Cent. FeO | Meth-od* | Magnetic Attraction per Gram at 800 Millamp. |          |
|--------------|-------------------|--|-----------------------------------|-----------------|--|---------------------------|-------|---------------|----------|--|----------|
|              |                   |  |                                   | No.             | Name                                   |                           |       |               |          | In Mg.                                       | Relative |
| 6,211        | 1,634             | Ferrous ammonium sulphate, $\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$ | J. T. Baker Chem. Co. (Lot 6,211) | .....           | .....                                  | 970.7                     | ± 25. | .....         | .....    | 2.72   | .....    |
|              | 1,638             |  |                                   | .....           | .....                                  | 1,004.                    | 21.6  | .....         | .....    | 2.71   | .....    |
|              | 1,645             |  |                                   | .....           | .....                                  | 1,035.7                   | 23.2  | .....         | .....    | 2.63   | .....    |
|              | 1,600             |  |                                   | .....           | .....                                  | 1,055.4                   | ± 22. | .....         | .....    | 2.63   | .....    |
| 4,913        | 1,601             | Ferrous sulphate, $\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$          | Do. "Special." (Lot 4,913)        | .....           | .....                                  | 1,158.1                   | ± 23. | .....         | .....    | 2.61   | .....    |
|              | 1,632             |  |                                   | .....           | .....                                  | 1,242.8                   | 27.   | .....         | .....    | 2.68   | 1.00     |
|              | 1,635             |  |                                   | 5" l            | Other red to Prussian red              | 1,014.                    | 22.2  | 0.015         | E        | 3.51   | 1.31     |
|              | 1,637             |  |                                   | 65" l           | Raisin black to dark slate-purple      | 1,020.                    | 22.   | 0.033†        | E        | 1.77   | 0.66     |
| 228          | 1,676             | Eisenoxyd zur Analyse mit Gerantieschein                             | Kahlbaum (Lot 4,478)              | 65" l           | Raisin black to dark slate-purple      | 1,107.0                   | ± 20. | 0.035         | E        | 2.01   | 0.75     |
|              | 1,676             |  |                                   | 69" l           | Anthracene purple to taupe brown       | 1,107.0                   | ± 20. | 0.035         | E        | 2.76   | 1.03     |
|              | 1,674             |  |                                   | 69" m           | Taupe brown                            | 1,024.0                   | ± 21. | 0.024         | E        | 3.02   | 1.13     |
|              | 1,674             |  |                                   | 69" m           | Taupe brown                            | 1,024.0                   | ± 21. | 0.024         | E        | 3.02   | 1.13     |
| 234          | 1,681             | Ferrous sulphate, $\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$          | Do. "Special." (Lot 4,913)        | 65" l           | Dark slate-purple to raisin black      | 1,039.9                   | ± 21. | 0.143         | E        | 3.82   | 1.43     |
|              | 1,677             |  |                                   | 65" k           | Dark heliotrope slate                  | 1,049.6                   | ± 21. | 0.050         | E        | 5.08   | 1.89     |
|              | 1,637             |  |                                   | 49" j           | Dark to blackish plumbeous             | 1,064.3                   | ± 23. | .....         | .....    | 6.92   | 2.58     |
|              | 1,637             |  |                                   | 49" j           | Do.                                    | 957.4                     | 23.   | .....         | .....    | 7.13   | 2.66     |
| 230          | 1,688             | Same heated in oxygen at 1,000°-1,100°                               | Hydrolysis solution               | 49" k           | Blackish plumbeous                     | 1,009.0                   | ± 23. | 0.082         | E        | 7.39   | 2.76     |
|              | 1,678             |  |                                   | 5' j            | Dragon-blood red to brick red          | 1,100.4                   | ± 20. | 0.000†        | E        | 7.77   | 2.90     |
|              | 1,691             |  |                                   | 5 k             | Morocco red                            | 651.9                     | ± 19. | .....         | .....    | 7.07   | 2.64     |
|              | 1,637             |  |                                   | 54" k           | Blackish plumbeous to dark violet-gray | 963.                      | 22.   | 0.094         | E        | 8.34   | 3.11     |
| 1,367        | 1,637             | Ferrous sulphate, $\text{FeSO}_4 \cdot 7\text{H}_2\text{O}$          | Kahlbaum nitrate, 260°            | 67" m           | Raisin black to taupe brown            | 905.9                     | ± 20. | 0.21          | P        | 17.47  | 6.51     |
|              | 1,692             |  |                                   | 67" m           | Raisin black to taupe brown            | 905.9                     | ± 20. | 0.21          | P        | 17.47  | 6.51     |
|              | 1,643             |  |                                   | 65" l           | Dark slate-purple to raisin black      | 442.7                     | 23.2  | 0.275         | E        | 22.2   | 8.28     |
|              | 1,638             |  |                                   | 49" k           | Blackish plumbeous                     | 434.                      | 21.6  | 0.23          | P        | 21.5   | 8.02     |

TABLE 5.—Ferroous Iron Content and Comparative Susceptibility of Artificial Oxides of Iron in Powder Form.—(Continued)

| Specimen No. | Magnetic Test No. | Designation of Specimen                         | Source  | Color of Powder |                                    | Wt. of Sample Tested, Mg. | Temp. | Per Cent. FeO | Meth-<br>od* | Magnetic Attraction per Gram at 800 Milliamp. |          |
|--------------|-------------------|---|---|-----------------|------------------------------------|---------------------------|-------|---------------|--------------|---|----------|
|              |                   |   |   | No.             | Name                               |                           |       |               |              | In Mg.  | Relative |
| 3526         | 1,673             | Ferrio oxide, ignited over blast lamp           | Ferrio nitrate, No. 1, from electrolytic iron               | 69''' j         | Vinaceous-ate to deep slaty brown  | 1,114.2                   | ± 19. | 0.097         | E            | 34.02   | 12.7     |
| 214          | 1,672             | Same, heated in oxygen and air at 1,100°-1,200° | .....   | 49''' k         | Blackish plumbecous                | 786.8                     | 19.1  | 0.109         | E            | 48.50   | 18.10    |
| 215          | 1,680             | Same, heated in oxygen at 1,100°                | .....   | 61''' k         | Dark plumbago slate                | 1,079.1                   | 20.   | 0.061         | E            | 41.58   | 15.53    |
| 229          | 1,678             | Ferrio oxide, ignited over blast lamp           | Do.   | 69''' l         | Anthracene purple to taupe brown   | 1,085.1                   | ± 21. | 0.035         | E            | 42.45   | 15.84    |
| 1,523        | 1,644             | Oxidised magnetite                              | Vacuum furnace, 1,200°                                      | .....           | .....                              | 634.4                     | 23.   | 0.30          | P            | 64.0  | 23.9     |
| 1,362        | 1,639             | Merck's ferric oxide, heated                    | Vacuum furnace, 1,100°                                      | .....           | .....                              | 68.3                      | 21.6  | 0.37          | P            | 76.0  | 28.4     |
| 1,347        | 1,640             | Merck's ferric oxide, heated                    | Do.   | .....           | .....                              | 150.8                     | 22.   | 0.55          | P            | 128.3   | 47.9     |
| 1,376        | 1,641             | Merck's ferric oxide, heated                    | Do.   | .....           | .....                              | 82.9                      | 21.5  | 0.77          | P            | 136.8   | 51.0     |
| 1,398        | 1,641             | Merck's ferric oxide, heated                    | Vacuum furnace, 1,200°                                      | .....           | .....                              | 63.5                      | 22.   | 1.17          | P            | 256.  | 95.5     |
| 1,340        | 1,642             | Merck's ferric oxide, heated                    | Do., 1,100°   | .....           | .....                              | 106.7                     | ± 23. | 1.96          | P            | 413.  | 153.2    |
| 242          | 1,683             | Oxidised magnetite                              | Mineville magnetite, quartz, and moist oxy-<br>gen, 470°    | 1'' k           | Mineral red                        | 1,330.4                   | ± 22. | 2.36          | E            | 494.  | 180.6    |
| 1,031        | 1,810             | Hematite scales                                 | Clunker from house-<br>heating furnace                      | 69''' k         | Deep slaty brown                   | 301.1                     | ± 18. | 6.48          | P            | 818.  | 305.1    |
| 1,539        | 1,834             | Oxidised magnetite                              | Vacuum furnace, 1,200°                                      | 61''' m         | Dull violet-black                  | 501.5                     | ± 19. | 4.65          | P            | 1,125.  | 420.     |
| 1,598        | 1,687             | Ferrio oxide from nitrate                       | Do.   | 49''' k         | Blackish plumbecous                | 932.6                     | ± 20. | 4.77          | P            | 1,185.  | 442.     |
| 240          | 1,682             | Oxidised magnetite                              | Mineville magnetite and moist oxygen, 460°                  | 69''' k         | Deep slaty brown                   | 625.9                     | ± 22. | 8.27          | P            | 1,179.  | 440.     |
| 224          | 1,675             | Oxidised magnetite                              | Bur. Stda. magnetite ore No. 29 and moist oxy-<br>gen, 400° | 5'' l           | Dark vinaceous-brown to seal brown | 969.3                     | ..... | 13.85         | P            | 1,878.  | 700.     |
| 1,548        | 1,832             | Oxidised magnetite                              | Vacuum furnace, 1,200°                                      | 67''' k         | Dark purplish gray                 | 1,067.2                   | ..... | 16.11         | P            | 2,948.  | 1,100.   |
| 1,551        | 1,833             | Oxidised magnetite                              | Do.   | 67''' l         | Dark to dusky pur-<br>plish gray   | 438.6                     | 17.   | 17.57         | P            | 4,856.  | 1,812.   |
| 1,043        | 1,657             | Black oxide                                     | Action of air on pasty iron amalgam                         | .....           | Black, mat                         | 479.2                     | 18.   | 66.1          | P            | 5,035.  | 1,878.   |
|              |                   |   |   |                 |                                    | 9.65                      | 21,   |               |              | 2,982.  | 1,113.   |

\* E, electrometric titration. P, Permanganate titration.

† Contains basic nitrate. Ignition loss 2.41 per cent.

‡ Not homogeneous; portions are as high as 0.043 per cent.

## NOTES ON TABLE 5

Samples Nos. 1,041 and 4,478a have been described and partial analyses given in *Journal of the American Chemical Society* (1916), **38**, 811.

Nos. 228, 226, 234, 230, and 229 were heated in open platinum crucibles, over the air-gas blast flame, which gives a temperature in such a crucible of about 1,100°. No. 352b was similarly heated in an Aupperle crucible.

Nos. 268, 274, 280, 214, and 215 were heated in platinum boats in a fused silica tube inside of a platinum-wound furnace. The apparatus and experiments are described in *Journal of the American Chemical Society* (1916), **38**, 1190.

Nos. 242, 240, and 224 were heated in platinum boats in the same tube furnace as the preceding; No. 242 for 45 hr., No. 240 for 20 hr. and No. 224 for 22 hr.

Nos. 231 and 247 were formed by heating a solution of the nitrate for several hours, the former in a sealed Jena combustion tube, the latter in a sealed silica tube, in a bomb furnace. We are indebted to Mr. E. G. Zies of this Laboratory for these heatings. The increase of temperature caused hydrolysis of the nitrate, precipitating the oxide, which settled to the bottom of the tube and was filtered off, washed, and dried.

Nos. 1,367, 1,521, 1,365, 1,523, 1,362, 1,347, 1,376, 1,398, 1,340, 1,539, 1,598, 1,548, and 1,551 were samples which had been used for the determination of dissociation pressures in the series  $\text{Fe}_2\text{O}_3$ - $\text{Fe}_3\text{O}_4$ , and are referred to in *Journal of the American Chemical Society* (1916), **38**, 824-826. All had been heated in platinum crucibles in the platinrhodium-tube vacuum furnace.

No. 1,072 was furnished to us through the kindness of Prof. C. E. Munroe, of George Washington University. The sample consists of well-developed platy specular hematite crystals, up to 1 cm. in diameter, which were taken from the pipes of a Deacon-process chlorine apparatus. These crystals and their mode of occurrence have been described by Prof. Munroe in *American Journal of Science* (1907), **24**, 485-486.

No. 1,031 consisted of glistening scales of hematite covering the surface of clinker from a house-heating furnace burning anthracite coal. Some unusual combination of circumstances, perhaps the association of a chloride with an ash high in iron and a moisture-laden air current, caused the deposition of hematite scales within the ash bed. Under the microscope the sample consisted of red scales. A dark residue, amounting to 11.5 per cent. of the original weight, remained after solution in HCl.

No. 1,043 was made by blowing air through a pasty iron amalgam, made by electrolyzing a ferrous sulphate solution with a mercury cathode. The product was shaken up with water, filtered, and washed with alcohol and ether.

were therefore made of a similar mixture which was ground for several hours in an automatic ore-grinding machine with an agate mortar, with the following results:

|  |                     |
|--|---------------------|
| Mineville magnetite No. 646, original..... | 30.84 per cent. FeO |
| Ground 2 hr. 20 min.....                   | 30.14 per cent. FeO |

The finely ground magnetite from the above test was mixed with three times its weight of quartz (between 100 and 200 mesh) and again ground in the automatic grinder:

|                                 |                    |
|---------------------------------|--------------------|
| Original mixture.....           | 7.53 per cent. FeO |
| Ground 5 hr. <sup>9</sup> ..... | 6.76 per cent. FeO |

The mortar was quite warm at the end of the test.

The results show that long-continued grinding causes oxidation of a part of the magnetite. These experiments confirm those of Mauzelius<sup>10</sup> and of Hillebrand<sup>11</sup> concerning the effect of such treatment on the ferrous-iron content of silicate rocks. They found, however, that 15 to 30 min. grinding is usually without serious effect.\* Our samples were all crushed as far as possible without grinding, and were then ground by hand for 5 to 15 min., with sifting and regrinding of residues. We believe, therefore, that none of the data of Tables 5 and 6 are affected by oxidation due to grinding.

#### EXPERIMENTAL RESULTS ON ARTIFICIAL AND NATURAL OXIDES

In Tables 5 and 6 we have assembled our data on the comparative susceptibility and the ferrous-iron content of artificial and natural oxides. Table 5 also includes tests on ferrous ammonium sulphate and ferrous sulphate, as a basis of comparison.

The first column of each table gives the reference number of the specimen. For the artificial oxides heated in the vacuum furnace these numbers correspond to those in our previous publications on the oxides of iron. The second column gives the reference number of the magnetic test. The third and fourth columns give the designation and description of the specimen and its source; further notes are given at the ends of the tables. The fifth column gives the color of the powdered substance when pressed flat in the glass testing-capsule. The color names and numbers are according to Ridgway.<sup>12</sup> In the sixth column is the weight of oxide

<sup>9</sup> The product with 6.76 per cent. FeO served as the original material for sample No. 242 of Table 5.

<sup>10</sup> R. Mauzelius: The Determination of Ferrous Iron in Rock Analysis. *Sveriges Geologiska Undersökning*, Årsbok (1907), No. 3.

<sup>11</sup> W. F. Hillebrand: The Influence of Fine Grinding on the Water and Ferrous-iron Content of Minerals and Rocks. *Journal of the American Chemical Society* (1908), 30, 1120-1131. The Analysis of Silicate and Carbonate Rocks. *U. S. Geological Survey, Bulletin* 422, (1910), 53.

<sup>12</sup> *Color Standards and Color Nomenclature*, Washington, 1912.

tested magnetically, and in the seventh the temperature. No correction was made for temperature variation, as the experimental error is in most cases as great as the correction to an average temperature. The eighth column contains the percentage of FeO on the total weight of sample, as determined analytically, followed (in Table 5 only) by a column indicating the method of analysis used. *E* indicates electrometric titration, *P* indicates titration with potassium permanganate. The last two columns give the magnetic attraction per gram of sample under the standard conditions, in milligrams as well as in multiples of the attraction on ferrous ammonium sulphate.<sup>13</sup>

### *Discussion of Results on Artificial Oxides*

The data of Table 5 are shown graphically in Fig. 3. In order not to crowd the samples low in FeO into an illegible jumble in the corner of the diagram, we have plotted the logarithm of the magnetic pull against the logarithm of the ferrous iron percentage. The curve of the quartz-magnetite mixtures of Table 4 is also included in Fig. 3.

The individual samples of Table 5 are arranged in the order of increasing magnetic susceptibility. From the table and from the curve of Fig. 3 it is evident that most of the artificial solid-solution samples have a magnetic attraction approximately proportional to their ferrous iron content. The possibility, suggested by the action of a hand magnet on the powders, that the attraction might increase much more rapidly than in proportion to the ferrous iron percentage, was not realized. From the standpoint of their magnetic susceptibility, these oxides are therefore indistinguishable from mixtures of magnetite and paramagnetic hematite. It is possible, however, that other magnetic properties will show a distinction between the two modes of occurrence of the ferrous iron.

Where the ferrous iron is present in some other form than in solid solution or in admixed magnetite, the corresponding magnetic attraction will fall below the mixture line. No. 1,031, for instance, which lies well below the line, undoubtedly contains considerable ferrous silicate.

When the percentage of FeO goes below about 0.2 per cent. the results are much less regular. There is a general drift of proportionality between FeO content and susceptibility, but other factors evidently become of importance. The variation in susceptibility is far too large to be accounted for by the FeO content or by any other of the known factors. The two samples made by hydrolysis of ferric nitrate form an

<sup>13</sup> Ferrous ammonium sulphate was adopted as the standard substance, not only because it is the purest and most easily reproducible salt of ferrous iron, but also because its magnetic susceptibility in the crystalline form has been carefully determined for all three crystallographic directions (W. Finke: *Annalen der Physik* (1910), **31**, 149-168).

extreme instance of this, as they contain no determinable ferrous iron, yet agree in having a susceptibility which is four times as high as that of the least magnetic oxide which contains determinable FeO. It is still possible, however, in this case, that some FeO existed in the presence of the excess of nitrate, and that it was oxidized during solution in HCl.

The four samples numbered 352b, 214, 215 and 229 are abnormally high in susceptibility for some as yet undiscovered reason. The cause evidently lies in the original ferric nitrate from which the samples were made. This nitrate was evaporated directly to dryness after solution of the electrolytic iron in nitric acid, and was not filtered or crystallized; a minute impurity in suspension, consisting of some undissolved and unoxidized substance like ferrosilicon, might explain the abnormal results. Oxides made from filtered and crystallized nitrate from the same iron are normal.

There is one striking exception to the general proportionality of magnetic susceptibility and ferrous-iron content, namely, the so-called "magnetic ferric oxide."<sup>14</sup> This can be easily prepared by the oxidation of precipitated magnetite at atmospheric temperatures, and is practically as ferromagnetic as magnetite itself. It loses its high susceptibility in a few minutes at 750°, and does not regain it on cooling. Slow oxidation of natural magnetite at 400° to 500° yields a non-magnetic oxide, as is evident from Nos. 242, 240 and 224, all of which have a susceptibility in normal agreement with their ferrous-iron content. Further experiments on the magnetic form of ferric oxide are in progress.

The oxide No. 1,043, which lies between Fe<sub>3</sub>O<sub>4</sub> and FeO in composition, is considerably less magnetic than Fe<sub>3</sub>O<sub>4</sub>. This is in qualitative agreement with the observations of Hilpert and Beyer.<sup>15</sup>

### *Colors of the Artificial Oxides*

The color of the powdered oxide was observed in most cases at the same time as the magnetic susceptibility. The color is stated in terms of Ridgway's standards.<sup>16</sup> A few words will serve to indicate the basis of the color system used by Ridgway. The pure spectrum colors and intermediate hues (including a series between violet and red) are numbered from 1 to 72. Tints, marked *a* to *g*, are made from these by admixture of white, and shades, marked *h* to *n*, by admixture of black. Each of these is further dulled by admixture of neutral gray, to form five series, indicated by accent marks from ' to ''''.

<sup>14</sup>S. Hilpert: Über Beziehungen zwischen chemischer Konstitution und magnetischen Eigenschaften bei Eisenverbindungen. *Berichte Deutschen Physikalische Gesellschaft* (1909), 11, 293-299.

<sup>15</sup>S. Hilpert and J. Beyer: Über Eisenoxyduloxyde und Eisenoxydul. *Berichte Deutschen Chemischen Gesellschaft* (1911), 44, 1608-1619.

<sup>16</sup>Robert Ridgway: *Color Standards and Color Nomenclature*, Washington, 1912.

Practically all the iron oxide hues in Table 5 lie in the range 65 to 72 and 1 to 5. No. 65 is a reddish red-violet and No. 5 is an orange orange-red. The oxide colors include no tints (tones lighter than the pure hues), and the shades (tones darker than pure hues) lie between *j* and *m* (59 to 87.5 per cent. black). There is only one example of a pure bright color, and only one example of the first degree of dullness (32 per cent. gray); the remainder are dulled by 58 to 95 per cent. gray.

The purer ferric oxides approach the pure red and orange-red colors, while those high in ferrous iron approach the darker shades of the same colors. The fineness of the powder has a much greater effect upon the color than the ferrous iron percentage. Samples 231 and 247, which are agglomerated colloids and extremely finely divided, furnish the two examples of the brighter colors referred to in the preceding paragraph. Equally pure oxides which have been sintered by heating at 1,100° or 1,200° go over toward the violet, although fine grinding brings their color back toward red.

These facts are in agreement with the microscopic characters of the oxides. Pure  $\text{Fe}_2\text{O}_3$  in thin flakes is deep red in color, and the color is darkened by solution of  $\text{Fe}_2\text{O}_4$ . The absorption is large, so that with increasing thickness the particles soon become opaque; the result is a change in color of the powder toward the violet.

#### *Discussion of Results on Natural Oxides*

The data of Table 6 are plotted in Fig. 4. As in Fig. 3, the scales of both magnetic attraction and percentage of  $\text{FeO}$  are logarithmic, to avoid the crowding of part of the data into one corner of the diagram. The quartz-magnetite curve of Fig. 3 is reproduced in Fig. 4.

The natural specimens in Table 6 are arranged in the order of increasing percentage of  $\text{FeO}$ . They were selected to give a wide variety of ferrous oxide percentages and a wide variety of modes of origin, rather than to show the variations in any particular locality or type of occurrence. Nearly all of them are clean and well-crystallized specimens. We hope later to devote a more detailed study to carefully located specimens from particular localities, as it is only in this manner that the data necessary for definite and detailed conclusions can be obtained. But there are certain general relations shown in Table 6 and Fig. 4 which should be pointed out.

1. If the natural solid solutions and mixtures of magnetite and hematite correspond in properties to the artificial, all of the points of Fig. 4 should be found on or below the quartz-magnetite mixture curve. With one exception, this is found to be the case. The single exception is the volcanic sublimate from Madeira; the cause of its high magnetism has not yet been found.



TABLE 6.—*Ferrous Iron Content and Comparative Susceptibility of Natural Oxides in Powder Form*

| Specimen No. | Magnetic Test No. | Designation of Specimen                   | Source                   | Color of Powder |                                   | Wt. of Sample Tested, Mg. | Temp. | Per Cent. FeO | Magnetic Attraction per Gram at 800 Millamp. |          |
|--------------|-------------------|---|--------------------------|-----------------|-----------------------------------|---------------------------|-------|---------------|--|----------|
|              |                   |   |                          | No.             | Name                              |                           |       |               | In Mg.                                       | Relative |
| 1,058        | 1,801             | Specular hematite                         | Lake Superior            | 5               | Morocco red                       | 1,040.6                   | ± 19. | 0.15°         | 14.88  | 5.55     |
| 1,090        | 1,803             | Hematite, twinned                         | Dognacks, Hungary        | 65'''           | Dark slate-purple                 | 714.8                     | ± 20. | 0.157†        | 20.0   | 7.46     |
| 1,011        | 1,814             | Specular hematite                         | Puy de Dome, France      | 69'''           | Anthracene purple to taupe brown  | 792.9                     | ± 21. | 0.28          | 18.28  | 6.82     |
| 1,026        | 1,808             | Specular hematite                         | Cumberland, England      | 65'''           | Dark slate-purple to raisin black | 1,416.3                   | ± 22. | 0.31          | 28.0   | 10.45    |
| 1,027        | 1,808             | Turgite portion of above                  | Altenberg, Saxony        | 5'              | Brick red                         | 964.8                     | ± 22. | 0.36          | 5.42   | 2.02     |
| 1,076        | 1,803             | Specular hematite                         | Altenberg, Saxony        | 65'''           | Dark slate-purple to raisin black | 1,211.0                   | ± 20. | 0.33          | 25.2   | 9.40     |
| 1,074        | 1,804             | Specular hematite                         | Island of Madeira        | 65'''           | Dark slate-purple                 | 1,645.6                   | ± 20. | 0.39          | 142.4  | 53.1     |
| 1,044        | 1,686             | Limonite                                  | Klaach Iron Works, Maine | 18'             | Ochraceous-tawny                  | 1,053.7                   | ± 22. | 0.62          | 3.06   | 1.14     |
| 1,685        | 1,685             | Miscellaneous hematite                    | Pennsylvania             | 67'''           | Raisin black to taupe brown       | 604.8                     | ± 19. | 0.45          | 15.09  | 5.63     |
| 1,028        | 1,684             | Martite                                   | Twin Peaks, Utah         | 65'''           | Raisin black to dark slate-purple | 1,152.6                   | ± 22. | 0.79          | 97.8   | 36.5     |
| 1,817        | 1,817             | Same, fractionated: (a) Magnetic fraction | .....                    | 69'''           | Anthracene purple to taupe brown  | 545.4                     | ± 22. | 1.13          | 193.6  | 72.2     |
| 1,816        | 1,816             | (b) Residual fraction                     | .....                    | 67'''           | Raisin black to taupe brown       | 1,029.5                   | ± 22. | 0.53          | 17.99  | 6.71     |
| 1,812        | 1,812             | Hematite, dodecahedral                    | Nova Scotia              | 69'''           | Taupe brown                       | 988.0                     | ± 21. | 1.44          | 191.2  | 71.3     |
| 1,806        | 1,806             | Hematite, crystalline                     | Dickerson, Md.           | 67'''           | Raisin black to taupe brown       | 437.6                     | ± 19. | 1.46          | 13.62  | 5.06     |
| 1,092        | 1,804             | Specular hematite                         | Guanaquabo, Mex.         | 69'''           | Anthracene purple to taupe brown  | 384.1                     | ± 22. | 1.73          | 38.5   | 14.37    |
| 1,804        | 1,804             | Iron ore: (a) Most magnetic fraction      | Juragua, Cuba            | 69'''           | Anthracene purple                 | 303.1                     | ± 21. | 2.59          | 495.   | 184.7    |
| 1,805        | 1,805             | (b) (10 per cent.) fraction               | .....                    | 69'''           | Anthracene purple to taupe brown  | 1,038.9                   | ± 21. | 0.67          | 58.2   | 21.7     |
| 1,805        | 1,805             | (c) (46 per cent.) fraction               | .....                    | 71'''           | Mars Violet                       | 1,052.1                   | ± 21. | 0.37          | 10.85  | 4.05     |
| 1,820        | 1,820             | (d) (64 per cent.) fraction               | .....                    | 69'''           | Anthracene purple to taupe brown  | 538.0                     | ± 21. | 3.13          | 582.   | 217.1    |
| 1,819        | 1,819             | Martite                                   | Cleveland Mine, Michigan | 69'''           | Deep slaty brown to aniline black | 535.0                     | ± 21. | 10.53         | 2,078.                                       | 775.     |
| 1,815        | 1,815             | Martite                                   | Twin Peaks, Utah         | 69'''           | Deep slaty brown to aniline black | 982.1                     | ± 22. | 10.37         | 2,690.                                       | 977.     |
| 1,094        | 1,821             | Iron ore                                  | Iron Hill, Colo.         | 69'''           | Taupe brown                       | 553.3                     | ± 20. | 12.96         | 2,739.                                       | 1,022.   |

(Continued on following page.)

TABLE 6.—*Ferrous Iron Content and Comparative Susceptibility of Natural Oxides in Powder Form.*—(Continued)

| Specimen No. | Magnetic Test No. | Designation of Specimen   | Source                | Color of Powder |                             | Wt. of Sample Tested, Mg. | Temp. | Per Cent. FeO | Magnetic Attraction per Gram at 800 Millamp. |          |
|--------------|-------------------|---|-----------------------|-----------------|-----------------------------|---------------------------|-------|---------------|--|----------|
|              |                   |   |                       | No.             | Name                        |                           |       |               | In Mg.                                       | Relative |
| 1,012        | 1,811             | Martite:<br>(a) Magnetic fraction (8 per cent.)<br>(b) Residual fraction (92 per cent.) | Minas Gerases, Brazil | 67"             | Raisin black to taupe brown | 217.1                     | ± 20. | 15.33         | 3,850.                                       | 1,437.   |
|              | 1,811             |   | .....                 | 71"             | Mars violet                 | 1,135.9                   | ± 20. | 0.65          | 34.63  | 12.92    |
| 1,084        | 1,807             | Hematite  | Tavetsch, Switzerland | 57"             | Blue-violet black           | 418.7                     | 22.   | 15.8          | 3,936.                                       | 1,468.   |
| 1,050        | 1,818             | Martite   | Woody, Cal.           | 69"             | Aniline black               | 509.1                     | ± 21. | 17.75         | 4,500.                                       | 1,678.   |
| 1,835        | 1,835             | Ilmenite  | Arendal, Norway       | 65"             | Dusky purplish gray         | 500.0                     | ..... | .....         | 84.8   | 31.6     |
| 3b           | 1,828             | Magnetite, 0.05 mm. grains and finer  | Mineville, N. Y.      | .....           | Black, lustrous             | 500.1                     | ± 19. | 29.63†        | 8,718.                                       | 3,253.   |
| 3b           | 1,825             | Magnetite, 0.16 mm. grains, average   | Do.                   | .....           | Black, lustrous             | 500.1                     | ± 20. | 29.66†        | 9,516.                                       | 3,551.   |

\* Permanganate titration. All of the analyses in this table were made by this method except those marked †.

† Electrometric titration.

## NOTES ON TABLE 6

1,011. Specular hematite, Puy de Dome. Foote Mineral Co., No. 825. A light brown trachytic tuff, impregnated with glistening hematite crystals up to 1 mm. in diameter. A portion was crushed and cleaned magnetically, and the crystals were washed with water. A dark gray residue, amounting to 6.3 per cent. of the original weight, remained after solution of the sample in HCl for analysis.

1,012. Martite, Minas Geraes, Brazil. Foote Mineral Co., No. 843. Loose, clean, bright octahedra, 1 to 4 mm. in diameter. Contains minute inclusions, probably rutile. Microscopic structure shows that the octahedra are pseudomorphs.

1,016. Micaceous hematite, unknown locality in Pennsylvania. Foote Mineral Co., No. 833. Radiating curving bundles of very thin leaves, associated with quartz. Magnetically homogeneous. Very slight residue, containing quartz, left after solution in HCl.

1,022. Martite, Twin Peaks, Millard Co., Utah. Ward's Natural Science Establishment. Intergrown octahedra, up to 25 mm. on edge. No evidence of any hematite-magnetite intergrowth visible under microscope. Powder is magnetically homogeneous, except for a slight residue containing apatite. Slight residue of silica left after solution in HCl.

1,026-1,027. Hematite, Cumberland, England, Foote Mineral Co., No. 821. Tabular crystals, associated with clear well-developed quartz crystals, on a red botryoidal base. The botryoidal portion has the microscopic characteristics of turgite. Powder contains several per cent. of calcite, whose high index indicates  $\text{FeCO}_3$  in solid solution.

1,028. Martite, Twin Peaks, Millard Co., Utah. Foote Mineral Co., No. 845. An altered quartzitic rock fragment covered on both sides with bright octahedra, 1 to 4 mm. in diameter. The octahedra are smaller on one side of the fragment than on the other. The larger octahedra contain some crystals of pyroxene. The larger crystals were powdered, and fractionated magnetically.

1,037. Hematite, Dickerson, Md. R. B. Sosman, No. 203. Intergrown with epidote and garnet. Occurs in Triassic calcareous shale, about 5 meters from west contact of a vertical diabase dike, about 100 meters wide. A slight residue remained after solution in HCl.

1,044. Limonite, Ktaadn Iron Works, Maine. G. P. Merrill, U. S. National Museum, No. 65,331. Occurs in loose porous form, frequently replacing the organic matter of logs, twigs, and leaves and forming "pseudomorphs" of decaying vegetation. The source of the iron is the pyrrhotite of nearby rocks, which weathers, giving rise to iron-bearing waters which flow into a bog and there deposit the limonite.

1,047. Martite, Cleveland Mine, Michigan. Foote Mineral Co., No. 232r. Bright octahedra, up to 1 mm. in diameter, thickly disseminated through fine-grained dark greenish-gray chlorite schist. Octahedra were separated from crushed rock by hand magnet.

1,050. Martite, near Woody, Kern Co., Cal. E. S. Larsen, No. W4. Loose-textured dark brown iron ore. A contact-metamorphic product, associated with granite. Sample is magnetically homogeneous, except for a slight residue of yellowish-brown limonite.

1,051. Martite, Iron Hill, Uncompahgre quadrangle, Gunnison Co., Colo. E. S. Larsen, No. U251. Octahedra, up to 15 mm. on edge, in loose brown matrix. Occurs on the Beaver Creek side of Iron Hill, in a martite-apatite dike about 7 meters wide, cutting limestone.

1,052. Martite, Nova Scotia. U. S. National Museum, No. 81,386. Dodecahedra, up to 4 mm. in diameter, in a reddish matrix, associated with crystalline quartz. The dodecahedral form is unusual.

1,058. Specular hematite, Lake Superior region. Foote Mineral Co., No. 232.

Occurs as velvety-appearing botryoidal lining in cavities in massive ore. A small amount of dark siliceous residue remained after solution in HCl.

1,074. Hematite, Island of Madeira. Collected by J. D. Dana. U. S. National Museum, No. 13,388. A fragment of reddish-brown basic igneous rock, encrusted with hematite scales up to 1 mm. in diameter. A volcanic sublimate. Incomplete solution in HCl leaves irregular hematite grains with projecting spikes. Solution in HCl leaves a residue of 2.6 per cent. of a silicate, containing inclusions.

1,076. Specular hematite, Altenberg, Saxony. Foote Mineral Co. Quartzitic rock containing stringers of black hematite, with free-growing crystals up to 2 mm. in diameter.

1,084. Specular hematite, Tavetsch, Switzerland. Foote Mineral Co. Hematite plates up to 2 mm. in diameter, on a joint face of a fragment of mica schist. Among the hematite crystals are some microscopic octahedra with rusty surfaces. About 3 per cent. siliceous residue remains after solution in HCl.

1,086. Specular hematite, Guanajuato, Mex. Foote Mineral Co. A fragment of rhyolite encrusted with hematite crystals up to 1 mm. in diameter. A bright-red residue, uniaxial and of high refractive index, probably a form of  $\text{TiO}_2$ , remains after solution in HCl.

1,090. Hematite, twinned, Dognacska, Hungary. Foote Mineral Co., No. 829. Crystals up to 3 mm. in diameter, associated with well-developed pyrite cubes, on massive magnetic ore. Crystals are twinned according to 1010.

1,092. Iron ore, Juragua, Cuba. W. Lindgren, No. 12 Juragua. From stock pile at Santiago Bay. Uniform blue-black glistening hematite ore, with few red partings. The ore is described by Lindgren and Ross in *Trans.* (1916), 53, 55-58.

1,094. Iron ore, Daiquiri, Cuba. W. Lindgren, No. 10 Daiquiri. From San Antonio lowest open cut. Dull gray ore with light yellow spots and partings. The ore is described in the article cited under 1,092, pp. 43-52.

2. If the  $\text{FeO}$  in a natural sample is not in solid solution in  $\text{Fe}_2\text{O}_3$ , or is not in the form of admixed magnetite, then its corresponding point in Fig. 4 should fall below the mixture line. The specimen which falls relatively farthest below this line is No. 1,044, the limonite from Maine, in which practically all of the ferrous iron is in some form not in solid solution in the  $\text{Fe}_2\text{O}_3$ , probably as  $\text{FeCO}_3$ ; this sample formed at atmospheric temperatures. No. 1,037, which formed in shale near a diabase dike, and No. 1,027 (the turgite portion of No. 1,026) come next in order. Both of these have probably been formed at lower temperatures than the highly ferrous specular hematites. About equally distant below the line is the Guanajuato hematite, No. 1,086, in which some of the  $\text{FeO}$  evidently loses its magnetism through combination with  $\text{TiO}_2$ . Various martites and specular hematites occur still nearer the mixture line.

3. The oxides higher in  $\text{FeO}$  lie much nearer the line than do those lower in  $\text{FeO}$ . The higher percentages of  $\text{FeO}$  in the former group, coupled with their more nearly normal magnetism, may indicate a higher temperature of formation of these oxides. Further experimental study of their conditions of formation and of their various magnetic properties will be necessary before final conclusions can be drawn concerning their origin.

4. Several of the specimens were fractionated magnetically (by the method described in the following paper), and the fractions were tested

and analyzed separately. In each case the less magnetic portion falls farther below the line than the more magnetic portion.

An interesting case of this kind is the Juragua iron ore, No. 1,092. The points representing the three fractions of the Juragua ore are connected by a broken line in Fig. 4. It is difficult to reconcile this graded

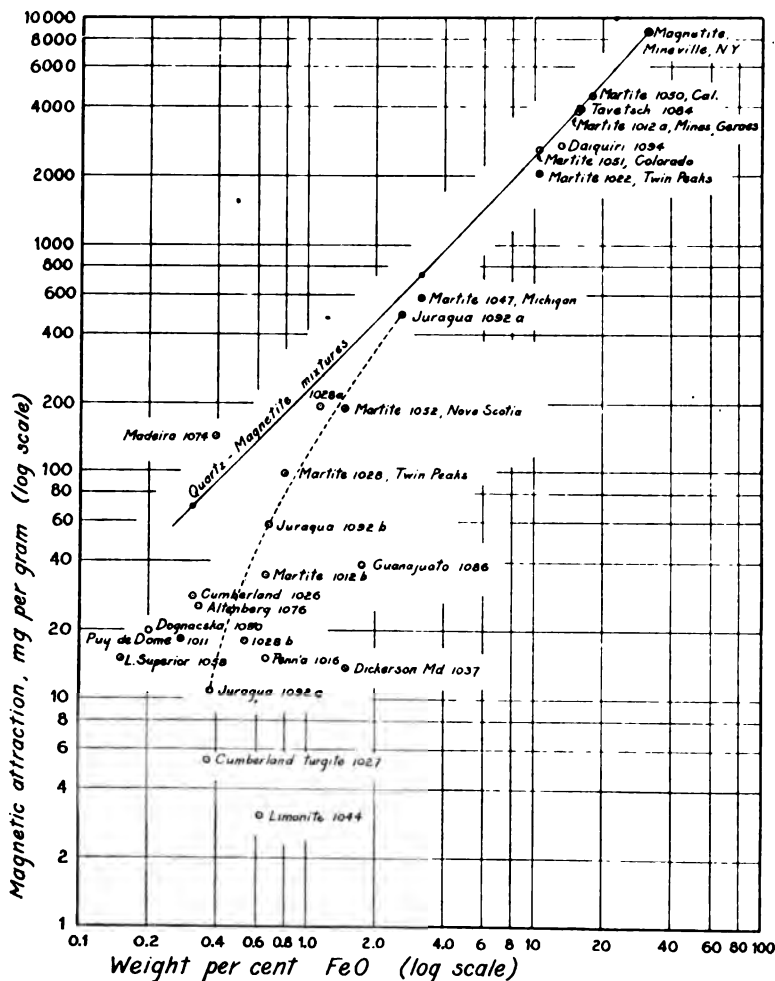


FIG. 4.—RELATION OF MAGNETIC ATTRACTION TO PERCENTAGE OF  $\text{FeO}$ , NATURAL OXIDES.

deviation of the successive fractions away from the mixture line with Lindgren and Ross's theory<sup>17</sup> of the origin of the South Cuban ores, according to which the ore was originally magnetite which has been oxidized by oxygen from the surface. If the ore were simply a mixture of

<sup>17</sup> W. Lindgren and C. P. Ross: *Trans.* (1916), **53**, 40-59.

hematite and unoxidized magnetite, all of the fractions should lie on the mixture line or deviate from it by a constant amount. But if the ore had been deposited by a vapor or solution of continually changing properties, so that the resulting product changed gradually in composition as the deposition (or alteration of previously deposited ore) proceeded, we might reasonably expect just the kind of gradation in magnetic properties that is actually found in the Juragua ore. This question will be considered further in the following paper on Zonal Growth in Hematite.

It should be noted that although both the Daiquiri and the Juragua ore specimens are "magnetic," in the usual sense, they are very far from being magnetites. Even the most magnetic fraction of the Juragua specimen contains only 2.59 per cent. FeO, whereas magnetite contains 31.03 per cent.

### *Colors of the Natural Oxides*

The colors of the powdered natural oxides cover approximately the same range as the colors of the artificial oxides (see Tables 5 and 6). Only two natural specimens, the Lake Superior hematite (1,058) and the Cumberland turgite (1,027) possess the pure colors characteristic of the hydrolyzed products in Table 5 (Nos. 231 and 247). Most of the other natural oxides are of the violet-red shades (65 to 71) characteristic of the artificial oxides which have been crystallized or sintered. Limonite 1,044, a hydrated oxide, is entirely different in color from the anhydrous oxides, being of a yellow-orange shade.

TABLE 7.—*Colors of Fractions of Elba Hematite Crystal*

| Portion  | Percentage of FeO | Color No.      | Color Name                             |
|----------|-------------------|----------------|--|
| <i>a</i> | 0.29              | 63''' <i>m</i> | Dull violet-black to raisin black      |
| <i>b</i> | .....             | 65''' <i>l</i> | Raisin black to dark slate-purple      |
| <i>c</i> | .....             | 67''' <i>m</i> | Raisin black to taupe brown            |
| <i>d</i> | 0.58              | 67''' <i>k</i> | Dark slate-purple to anthracene purple |
| <i>e</i> | .....             | 69''' <i>l</i> | Anthracene purple to taupe brown       |
| <i>f</i> | 4.24              | 67''' <i>m</i> | Raisin black to taupe brown            |
| <i>g</i> | .....             | 65''' <i>l</i> | Raisin black to dark slate-purple      |
| <i>h</i> | .....             | 69''' <i>l</i> | Anthracene purple to taupe brown       |
| <i>i</i> | 6.76              | 67''' <i>m</i> | Raisin black to taupe brown            |
| <i>j</i> | .....             | 65''' <i>m</i> | Raisin black                           |
| <i>k</i> | 3.31              | 63''' <i>m</i> | Dull violet-black to raisin black      |
| <i>l</i> | .....             | 65''' <i>m</i> | Raisin black                           |
| <i>m</i> | .....             | 67''' <i>m</i> | Raisin black to taupe brown            |

The fact that fineness of grain, rather than chemical composition, has the predominating influence upon the color, is shown in Table 7, in which are given the colors of the powders of the various fractions of an Elba hematite crystal (which will be further described in the paper which follows). The specimens were all crushed to pass a 100-mesh sieve (open-

ing 0.140 mm.). Although the ferrous iron in these fractions covers the wide range from about 0.2 to 6.76 per cent. FeO, nevertheless the hues all lie within the range 63 to 69, the shades are all within the range *k* to *m*, and the dullness of all the samples is the same (expressed by "", or 77 per cent. gray). These colors were measured on different days, and without comparison of one powder with another, but only by direct comparison with the color standards.

### MARTITE

The mineral martite is generally considered to be a pseudomorph of hematite after magnetite. It frequently has the composition of nearly pure  $\text{Fe}_2\text{O}_3$ , while retaining the isometric (usually octahedral) external form of magnetite. But many specimens, like several of those in Table 6, for example, contain large percentages of FeO, and are magnetically homogeneous, suggesting the possibility that they may be true solid solutions of  $\text{Fe}_3\text{O}_4$  in  $\text{Fe}_2\text{O}_3$ .

These martites are found in fact to be both pseudomorph and solid solution at the same time. Their fibrous or granular character shows that they are real pseudomorphs, but the fibers or granules themselves appear to consist of a solid solution of  $\text{Fe}_3\text{O}_4$  in  $\text{Fe}_2\text{O}_3$ .

The following descriptions<sup>18</sup> of the microscopic characters of three natural martites whose ferrous iron content and magnetic susceptibility have been given in Table 6 will illustrate the properties of this mineral.

No. 1,012. Martite from Minas Geraes, Brazil. The octahedra are fine-grained aggregates of hematite-colored, strongly doubly refracting granules, and are therefore pseudomorphic.

No. 1,047. Martite from Cleveland Mine, Michigan. Aggregates similar to those of No. 1,012.

No. 1,022. Martite from Twin Peaks, Utah. In powder the grains appear to be single units optically, but thin sections show that these grains are fragments of larger platy crystals which form roughly radiating aggregates within the octahedral crystals. These aggregates occupy the surfaces as well as the interiors of the octahedra. This, also, is a case of pseudomorphism. The grains in the powder are found to be uniaxial and negative.  $\epsilon$  for wave-length  $700\mu$  was observed to be 2.635. The double refraction for this wave-length (as determined by observing the maximum number of orders of interference colors around grains about 0.01 mm. thick) is about 0.230.

Although this martite is optically somewhat similar to hematite, the refractive indices are much lower. The lowering can be due only partly to FeO, if at all, for the martite is about as light-colored as pure hematite. The refractive indices are intermediate between hematite and the fibrous water-containing form of  $\text{Fe}_2\text{O}_3$  known as turgite, in which the usual

<sup>18</sup> Microscopic Examinations by H. E. Merwin.

values of the refractive indices are:  $\epsilon_{Li} = 2.37$  to  $2.47$  and  $\omega_{Li} = 2.48$  to  $2.57$ .

Pleochroism is not noticeable, although an apparent stronger absorption of  $\omega$ , caused chiefly or entirely by reflection from the irregular surfaces of grains immersed in a substance having the refractive index  $\epsilon$ , was observed.

### *Origin of Martite*

The solid solutions that would be formed at temperatures approaching atmospheric, and by water solutions that were oxidizing rather than reducing in character, would be high in ferric iron and low in ferrous. The alteration of magnetite at a relatively low temperature and under oxidizing conditions should therefore yield a martite high in ferric iron. On the other hand, formation from magnetite at a relatively high temperature is strongly suggested by the properties of those martites that are high in ferrous iron, and which are at the same time homogeneous.

There is another possible criterion which can be applied to martite. The oxidation of artificial magnetite at ordinary temperatures in the laboratory yields a ferric oxide which is practically as ferromagnetic as magnetite itself (see page 921). This high magnetic susceptibility is lost in a few minutes at  $750^\circ$ , but there is no evidence as yet to show that it disappears or even diminishes at atmospheric temperatures. On the contrary, we have found one undoubted natural example of it which must have persisted for a period of time measurable in geologic terms. It appears probable,<sup>19</sup> also, that the oxidation of natural crystalline magnetite at low temperatures yields a magnetic  $\text{Fe}_2\text{O}_3$ . If, therefore, the magnetism of a given natural iron oxide is found to be in normal agreement with its content of ferrous iron, the natural inference is that either (1) this percentage of ferrous iron is original, or (2) the oxide has been subjected to a temperature high enough to have brought its magnetic susceptibility down to the normal value corresponding to its ferrous iron content. How high this temperature would need to be under geological conditions remains to be determined.

What was the agent and the process of the undoubted alteration of many magnetites into martites? In considering this question, we are brought face to face with the problem of the permeability of minerals to liquids or vapors. The conversion of magnetite to martite is so complete and uniform in many cases, yielding "crystals" which have no core or other evidence of alteration from the outside inward, that one is strongly

<sup>19</sup> War emergencies have temporarily interrupted the work and have prevented our obtaining complete proof of the oxidation of natural crystalline magnetite to magnetic  $\text{Fe}_2\text{O}_3$ . Magnetite is so resistant to oxidizing agents that it is difficult to bring the  $\text{FeO}$  down to zero at a temperature which will not destroy the magnetism of the  $\text{Fe}_2\text{O}_3$ . We have, however, succeeded in oxidizing Mineville magnetite to an oxide, which, although it still contains an appreciable percentage of  $\text{FeO}$ , is much more magnetic than a normal oxide of the same composition.



tempted to believe that the altering solutions passed straight through the body of the magnetite. We know that metallic iron at a red heat is permeable to carbon monoxide, that hot platinum is permeable to hydrogen, and that hot silver is permeable to oxygen. Is there a temperature at which magnetite is permeable to water vapor or to water solutions of oxidizing agents? This question of permeability need not be confined to magnetite, for petrographic evidence is continually suggesting that solutions or vapors have at some time passed directly through minerals which under atmospheric conditions are impermeable.

### SUMMARY

The percentage of ferrous iron and the relative magnetic susceptibility in powder form have been determined on a number of artificial and natural oxides of iron. Artificial oxides made at 1,100° and 1,200° consist of a solid solution of  $\text{Fe}_3\text{O}_4$  in  $\text{Fe}_2\text{O}_3$ . Their relative magnetic susceptibility is approximately proportional to their percentage of  $\text{FeO}$ , from  $\text{Fe}_2\text{O}_3$  over to  $\text{Fe}_3\text{O}_4$ . The deviations may be partly accounted for by the effect of various factors, of which the fineness of grain of the powdered oxide is the most important, especially in the case of the more ferromagnetic members of the series. The colors of the powdered oxides depend both on their chemical composition and on their physical constitution, especially the fineness of grain.

In addition to the oxides whose susceptibility depends upon their content of  $\text{FeO}$ , there exists also a highly ferromagnetic form of  $\text{Fe}_2\text{O}_3$ , which appears to be rare in natural occurrence.

The natural iron oxide minerals are similar to the artificial in being in many cases solid solutions of  $\text{Fe}_3\text{O}_4$  in  $\text{Fe}_2\text{O}_3$ . Others are mixtures of  $\text{Fe}_3\text{O}_4$  and  $\text{Fe}_2\text{O}_3$ . If the ferrous iron is not in solid solution or in magnetite admixture, the magnetic susceptibility falls below the normal.

Some natural oxides can be magnetically fractionated; in these cases the less magnetic portions are found to deviate more widely from normal than the more magnetic. The cause of this deviation is not yet entirely clear.

Martite is a pseudomorph after magnetite, but its constituent granules or fibers consist usually of a solid solution of  $\text{Fe}_3\text{O}_4$  in  $\text{Fe}_2\text{O}_3$ . The ferrous iron content and the magnetic susceptibility of the specimens examined suggest that they have been produced at temperatures considerably higher than atmospheric.

We wish to acknowledge our indebtedness to Messrs. Merrill, Wherry, Larsen, Lindgren, and Munroe for specimens of oxides of iron; to Mr. C. W. Burrows of the U. S. Bureau of Standards for suggestions concerning the magnetic tests; and to our colleagues of the Geophysical Laboratory, especially to Mr. Merwin for microscopic examinations and to Mr. H. S. Roberts for the electrometric titrations.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Zonal Growth in Hematite, and Its Bearing on the Origin of Certain Iron Ores

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(St. Louis Meeting, October, 1917)

### INTRODUCTION

WE have shown in the preceding paper that practically all natural oxides of iron contain a determinable percentage of ferrous iron, and in many cases the percentage approaches that in magnetite itself. Some of these oxides are true homogeneous solid solutions. Others are not homogeneous, and can be separated magnetically into fractions. The nature and cause of this inhomogeneity in certain special cases form the subject of the present paper.

### MAGNETIC FRACTIONATION OF OXIDES OF IRON

The magnetic fractionation of a sample of oxide can be conveniently carried out by powdering the material very fine, and exposing it at various distances from the poles of an electromagnet. The magnet that we used was specially designed<sup>1</sup> to give a strong field together with a steep gradient outside of the poles. It is set in a frame, with the pole pieces directed downward. A glass plate can be adjusted at various distances beneath the poles. If the fine iron oxide powder is placed just beneath the glass plate, successive portions of the powder are picked up by the magnetic attraction and are caught against the glass plate, the most magnetic portions being taken up at the greatest distance from the poles. Some natural oxides can thus be separated into several fractions of different magnetic susceptibilities.

Among other natural iron oxides, we examined the powder obtained by crushing some crystals of hematite from Elba.<sup>2</sup> We found that

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\* Geophysical Laboratory, Carnegie Institution of Washington.

<sup>1</sup> This magnet was designed by C. W. Burrows, of the Bureau of Standards, for F. E. Wright of this Laboratory.

<sup>2</sup> Elba hematite crystals are well known in museums for their large size and excellent development. It is said that a great many of those now in American collections were found in shipments of Elba ore which came into the port of Philadelphia up to the year 1895.

the powder contained a notable percentage of ferrous iron, and also that it could be fractionated magnetically in the manner described above. The possibility of fractionation shows that the crystal is not homogeneous, as would be the case if it were a uniform solid solution throughout. It

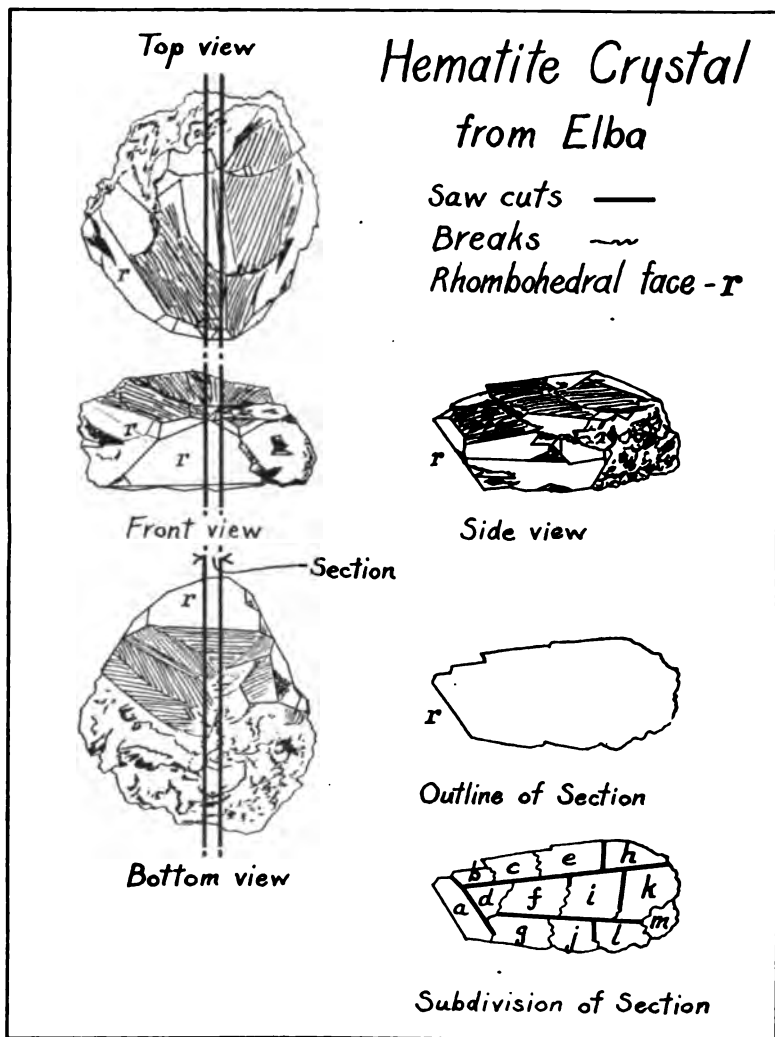


FIG. 1.—SECTIONING OF ELBA HEMATITE CRYSTAL FOR MAGNETIC TESTS.

is immediately evident that there may be two possible explanations of this inhomogeneity:

1. The crystal may be uniform in gross composition, but at the same time heterogeneous, to the extent that it consists of an intimate mixture

of very small crystals of hematite and magnetite, or of two or more other intermediate oxides differing in their percentage of FeO and in their magnetic susceptibility; or

2. The percentage of FeO and the magnetic susceptibility may vary in some continuous manner along the axes of the crystal.

We sought to decide between these alternatives by cutting a thin slice through the middle of a large Elba crystal, and then cutting this slice up into small fragments. Each of these was tested separately, and the results correlated with the position of the fragment in the crystal.

The shape of the crystal and the location of the sections are shown in Fig. 1, drawn approximately to scale, and oriented by a comparison of the faces with crystallographic descriptions of the crystals in Dana's *System of Mineralogy*.

### *Apparatus and Method*

Each of the lettered fragments of Fig. 1 was powdered to pass a 100-mesh sieve (opening 0.140 mm.), and the powder was tested magnetically by measurement of the pull in milligrams exerted on a known weight of it by the field of an electromagnet, at a definite value of the magnetizing current. Absolute measurements of the magnetic constants on such small and irregular samples are out of the question, but comparative results are readily obtained by this method. Description and tests of the apparatus will be found in the preceding paper.

### *Experimental Results*

In Table 1 are given the magnetic attractions in milligrams per gram of oxide at the maximum magnetizing current of 800 milliamp., for each of the subdivisions of the section of the crystal. The letters correspond to those in Fig. 1. The attractions at the other values of magnetizing current would give no additional information for our present purpose, and are omitted.

Some idea of the susceptibility in absolute measure of the less magnetic portions may be obtained from the fact that ferrous ammonium sulphate,  $\text{FeSO}_4 \cdot (\text{NH}_4)_2\text{SO}_4 \cdot 6\text{H}_2\text{O}$ , under the same conditions, gives an attraction of 2.68 mg. per gram.

The percentage of ferrous oxide, FeO, as determined by the chemical analysis of several of the portions of which there was a sufficient quantity available, is given in the fourth column of the table. The method of analysis was the same as that used in previous work on the oxides of iron, and has been described elsewhere.<sup>3</sup>

<sup>3</sup> J. C. Hostetter: *Journal of the American Chemical Society* (1916), **38**, 821.

The data for those portions that were tested both magnetically and chemically are plotted in Fig. 2. The figure also contains a curve showing the relation between the magnetic attraction at 800 milliamp. and the percentage of FeO in mixtures of magnetite and quartz.

The magnetic data, supported by the chemical analyses, show that the hematite crystal from Elba varies in composition from the outside

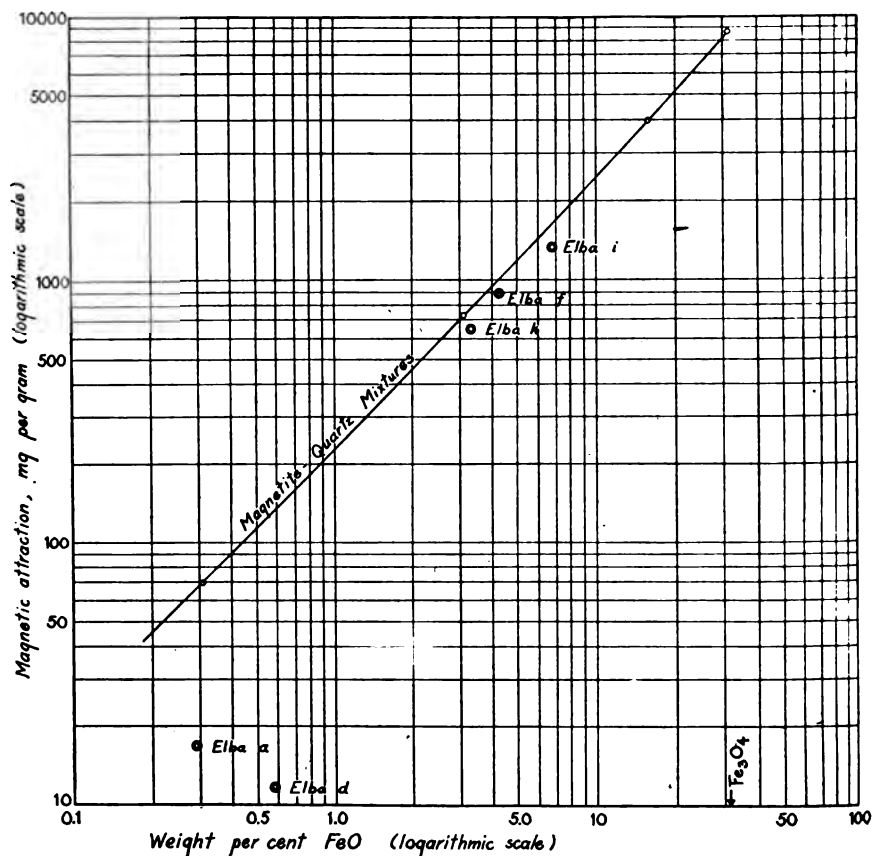


FIG. 2.—RELATION OF MAGNETIC ATTRACTION TO PERCENTAGE OF FeO.

to the center, and, in particular, from the "tip," which is bounded by good faces and evidently grew freely without interference, toward the "base," which is irregular as if broken out of the wall of a cavity. This variation of composition is shown graphically in Fig. 3, in which "contours" are drawn in the plane of the section, each line representing in a general way the locus of points of the same magnetic susceptibility, and hence of the same composition. The dotted lines are drawn to indicate the

probable extension of these lines of constant composition from the base of the crystal into the matrix of iron ore from which it was broken.<sup>4</sup>

TABLE 1.—*Magnetic Attraction on Various Portions of Elba Hematite Crystal*

| Portion | Weight of Sample Tested | Attraction at 800 Milliamp. | Percentage of FeO |
|---------|-------------------------|-----------------------------|-------------------|
|         | Mg.                     | Mg. per Gram                |                   |
| a       | 634.4                   | 16.8                        | 0.29              |
| b       | 185.5                   | 17.6                        |                   |
| c       | 325.1                   | 14.2                        |                   |
| d       | 328.1                   | 11.7                        | 0.58              |
| e       | 508.2                   | 23.3                        |                   |
| h       | 493.1                   | 80.9                        |                   |
| g       | 238.9                   | 75.9                        |                   |
| j       | 230.6                   | 138.4                       |                   |
| l       | 349.5                   | 222.4                       |                   |
| f       | 483.3                   | 897.                        | 4.24              |
| i       | 578.4                   | 1,318.                      | 6.76              |
|         | 577.1                   | 1,327.*                     |                   |
| k       | 847.8                   | 654.                        | 3.31              |
| m       | 126.3                   | 874.                        |                   |

\* Redetermination after removing and mixing the powder, to show degree of reproducibility of measurements on the most ferromagnetic member of the series.

### CAUSE OF THE ZONAL GROWTH

It is of interest to inquire what physical and chemical conditions could have produced such a zonal growth.

The origin of the Elba iron ores was for many years in controversy. Vom Rath, Cocchi, and Reyer, among the earlier geologists, believed the deposits to have been originally sedimentary. Later work, especially that of Lotti,<sup>5</sup> has shown that the ores are associated with post-Eocene salic igneous intrusions, and that they belong to the class of contact deposits.

The precise nature of the process by which these and other similar deposits were formed is of course difficult or impossible to state on the

<sup>4</sup> Measurements of the relation of physical properties, such as elasticity and magnetic susceptibility, to the crystal axes have frequently been made on large hematite crystals, usually on the assumption that the crystals were homogeneous. The present results show that the possible presence of zonal growth should always be taken into account in such measurements before conclusions are drawn as to the relation of properties to crystal axes. This is especially true of magnetic properties, on which a small percentage of FeO has so large an effect.

<sup>5</sup> B. Lotti: *Descrizione geologica dell'Isola d'Elba. Mem. descr. carta geol. Italia* (1886), 2, 193-235.

basis of field evidence alone. The explanation must be given in such general terms that no quantitative deduction can be drawn. The Italian mining geologist Cortese makes the following statement concerning the ores: "They are due to endogenous emanations of fluids and vapors at high temperature and under great pressure, which bore along with them the oxides of iron that were ultimately deposited in the hollows of the older rocks."

Let us put this statement into more precise terms, in order to find what deductions can be made from it and how they agree with the facts. Let us assume, for example, that the ores were deposited from a vapor which escaped from the igneous rock, and which contained iron, chlorine, hydrogen, and oxygen, at a temperature somewhere between  $100^{\circ}$  and  $700^{\circ}$ , and a pressure of 300 atmospheres or less. Now a mixture of the compounds of iron, chlorine, oxygen and hydrogen under these conditions is a

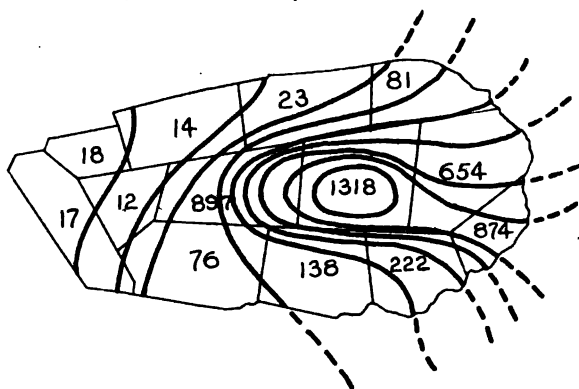


FIG. 3.—DISTRIBUTION OF FeO IN CROSS-SECTION OF ELBA CRYSTAL.

four-component chemical system. The phase rule and the chemical properties of the system enable us to draw certain conclusions concerning its behavior.

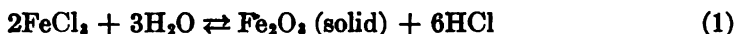
Two solid oxides, both being members of the hematite-magnetite series, cannot deposit out of such a vapor simultaneously, because the oxygen pressure falls continuously throughout the series  $\text{Fe}_2\text{O}_3\text{-Fe}_3\text{O}_4$ . Since the oxygen pressure in the vapor has a definite value, there can be only one oxide of this series in equilibrium with the vapor.

As only one oxide is depositing, the system consists of two phases: solid and vapor. If the original composition of the vapor is fixed, then the ratio of chlorine to hydrogen in the vapor must remain constant, since only iron and oxygen are depositing. The system is therefore ternary; that is to say, there may be a range of pressures, a range of temperatures, and a range of concentrations of one of the components with



respect to the others, within which the deposition of the oxide may be taking place.

Disregarding for the present the doubled molecules  $\text{Fe}_2\text{Cl}_6$  and  $\text{Fe}_2\text{Cl}_4$  (which dissociate as the temperature rises), we may consider the deposition of oxides from the supposed vapor as based upon the following reactions:



As we have shown experimentally,  $\text{FeO}$  and  $\text{Fe}_2\text{O}_3$  form a solid solution from  $\text{Fe}_2\text{O}_3$  toward  $\text{Fe}_3\text{O}_4$ . This solid solution acts as a single solid phase. The reactions (1) and (2) can therefore occur simultaneously, and yet, as shown above, a single oxide phase of varying composition can be depositing through a range of temperatures, a range of pressures, or a range of concentrations. Under conditions of complete equilibrium, the composition of this oxide should change as a whole, as the pressure, temperature, or composition of the vapor changed. Actually, however, the interior of crystals already deposited might be protected against further reaction by the continued deposition of more oxide on the surface. The result would be a zoned crystal like the Elba crystal which we have just described. Similar zonal growths are very common in minerals which form solid solution series, as, for instance, in the lime-soda feldspars.

We have assumed, for the sake of definiteness in discussion, that the iron was carried in a vapor containing iron, chlorine, hydrogen, and oxygen. It is not, however, necessary for the production of the zonal crystals that they should have been deposited from this particular vapor, or, in fact, from any vapor. The general principles remain the same, even though the iron were carried in a vapor of some other composition, or in a liquid solution.

### *Conditions Indicated by the Zonal Growth*

Two possible conditions of growth of the zoned crystals must be distinguished at this point:

1. The crystals may have formed in an open fissure through which vapor or solution was continually passing, supplied from a distant source.
2. The crystals may have grown in a closed space to which vapor or solution was not continuously supplied.

In either case, as we have shown above, a varying temperature, pressure, and concentration of the vapor is compatible with the deposition of the mixed oxide. But the phase rule gives no information as to how the percentage of ferrous iron in the crystal will vary with a given change in pressure, temperature, or concentration. A complete quantitative statement can only be made if we know (1) the equilibrium constants

of the reactions that take place, (2) the heat capacities and heats of formation of the compounds that take part in the reactions, and (3) the volume changes that take place in the reactions. This information is available only in part. We have good data,<sup>6</sup> for instance, on the reaction



which is one of the reactions that would occur in the vapor postulated above. Other reactions which would occur, and whose constants determine the equilibrium, are:



Until further experimental data on the reactions (1) to (5) are obtained, it will not be possible to make quantitative deductions regarding the effect of the different variables upon the ferrous-ferric ratio in the solid oxide. Certain qualitative deductions are, however, possible. The apparently continuous gradation of ferrous iron in the zoned crystal indicates that there was a continuous change in the temperature, pressure, and concentration relations of the vapor. If the crystals grew in a closed space, cut off from fresh supplies, it appears probable that falling temperature and the removal of ferrous and ferric iron from solution would determine the equilibrium. If they grew in an open fissure, continually supplied by vapor or solution from the igneous source, the zoning might represent (1) decreasing temperature of the source, (2) decreasing pressure in the supply, or (3) changing composition of the vapor or solution as the supply of constituents capable of escaping became exhausted.

It is an interesting fact that the zoning of the crystals is duplicated in the orebodies themselves. In general, the magnetite-bearing ore and the specularite (which always carries ferrous iron) are found in the deeper parts of the deposits, while the compact hematite is found in the central and upper portions. The hematite frequently passes into limonite in the upper portion, especially where the ores terminate against limestone; whether this limonite is the product of hypogene or supergene solutions must remain for the present an open question. The fact remains that there is a decrease in ferrous iron in passing upward through the orebodies, a condition that may well have resulted from the same series of changes in temperature, pressure and concentration as those which gave rise to the zoned crystals.

This association of the oxides is not peculiar to the Elba ores, but has been reported in other contact deposits. The more ferrous oxide, magnetite, is found nearer the igneous source, while the more ferric

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<sup>6</sup> G. N. Lewis: Equilibrium in the Deacon Process. *Journal of the American Chemical Society* (1906), **28**, 1380-1395.

oxide, hematite, is found farther away. At Cornwall, Pa., for instance, "we have a foot wall of diabase, a hanging wall of limestone, with the [magnetite] ore between. . . . We also find a small amount of hematite mostly at or near the contact with the limestone."<sup>7</sup> In the Firmeza district, Cuba (Juragua Iron Co. mines), "as the ore becomes less massive toward the margin of the deposit the proportion of specularite to magnetite increases."<sup>8</sup> "There is an increase in hematite in the higher levels of the mines."<sup>9</sup> In the Daiquiri district of southern Cuba, also, "hematite appears to be more abundant close to the surface than in depth."<sup>10</sup> "The fact that there is this definite arrangement of the increase of hematite above and increase of magnetite in depth is too consistent and widespread to be fortuitous when you consider that the Daiquiri mines at which Dr. Lindgren observed it, and the Juragua and the Ocania mines, stretch over a distance of about 15 miles, and in each case the same arrangement seems to hold good."<sup>11</sup>

Lindgren and Ross interpret the distribution of oxides in the Cuban deposits as evidence of the oxidation of a magnetite ore by oxygen-bearing waters from the surface. Roesler, Graton, and others believe that the distribution is either original or produced by the action of heated solutions from below upon the original oxide. In the preceding article we have noted that the magnetic properties of our specimen of the Juragua ore are not in accord with Lindgren's interpretation. It is only necessary to add that, as we have shown above, the zonal distribution of ferrous iron in such a deposit is readily accounted for by the same considerations as to changing temperature, pressure, and concentration of a vapor or solution as were used to account for the zonal distribution of ferrous iron in the Elba crystal. But we should not exclude the possibility that some surface oxidation has been superposed on the previously existing ferric percentage, and in addition it must be remembered that oxidation of pyrite may have produced a little additional ferric oxide in the form of limonite.

The possible action of an acid vapor or solution, such as our assumed iron-chlorine-oxygen-hydrogen vapor, upon calcium carbonate is of special interest in connection with the Elba deposits. The ores terminate in many places against limestone, and have evidently replaced large bodies of this rock. This is shown clearly by the residual blocks of limestone found in unaltered stratigraphic position within the ore,<sup>12</sup> and by the

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<sup>7</sup> H. Souder: *Discussion in Trans.* (1916), 53, 66.

<sup>8</sup> M. Roesler: *Bull.* (Oct., 1916), 1811.

<sup>9</sup> M. Roesler: *Op. cit.*, 1815.

<sup>10</sup> W. Lindgren and C. P. Ross: *Trans.* (1916), 53, 55.

<sup>11</sup> M. Roesler: *Discussion in Trans.* (1916), 53, 66.

<sup>12</sup> B. Lotti: *Geol. Föt. Förh.* (1891), 13, 599-603.

fact that even the original caverns of the limestone have sometimes been preserved in the limonitic portions of the orebodies.<sup>13</sup>

The principal reaction occurring at the contact of a vapor, such as we have postulated, with the limestone would be:



From reactions (1) and (2) it is evident that the removal of HCl from the vapor by reaction with  $\text{CaCO}_3$  would cause the deposition of oxides of iron, while the  $\text{CaCl}_2$  formed would pass off in solution. The lime of the  $\text{CaCl}_2$  would probably not be redeposited elsewhere, on account of the great solubility of  $\text{CaCl}_2$  and the absence of any precipitating agent. The  $\text{CO}_2$  would mostly escape as gas, unless water was in large excess, in which case considerable lime would be taken into solution as bicarbonate. According to Capacci,<sup>14</sup> neither gypsum (calcium sulphate) nor travertine (calcium carbonate) are found in association with the Elba ores; he believes that the former fact stands in the way of any theory involving sulphate solutions, while the latter suggests the absence of large amounts of carbonated waters.

A suggestion as to the conditions under which specular crystals such as those from Elba may be deposited can be obtained from some excellently developed artificial hematite crystals which were kindly furnished us by Prof. Munroe of George Washington University. He obtained them some years ago from a Deacon-process apparatus in Philadelphia.<sup>15</sup> The Deacon process depends upon the oxidation of hydrochloric acid by atmospheric oxygen, and is usually operated at a temperature of about 400°. The crystals in question were taken from the inside of the digester pipes, after a shutdown which was due, not to corrosion from within, but to the burning through of the pipes by  $\text{SO}_2$ -bearing furnace gases from without. The iron of the hematite crystals may have come in as an impurity in the hydrochloric acid, and was probably deposited according to reactions (1) and (2), page 938. We found by analysis that these crystals contain 0.21 per cent.  $\text{FeO}$ , which is about the same as the tip of the zoned Elba crystal. It should be remembered, however, that the ferrous-iron-bearing Deacon-process crystals were deposited out of a strongly oxidizing vapor; the tip of the Elba crystal might have grown at a very different temperature and pressure if the composition of the vapor from which it deposited was very different from that in the Deacon process.

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<sup>13</sup> C. Capacci: The Iron Mines of the Island of Elba. *Journal of the Iron & Steel Institute* (1911), **84**, 412-450.

<sup>14</sup> C. Capacci: *Op. cit.*, 442.

<sup>15</sup> C. E. Munroe: Artificial Hematite Crystals. *American Journal of Science* (1907), **24**, 485-486.

## SUMMARY

The powdered oxide from certain crystals of hematite from Elba contains considerable FeO, and can also be fractionated magnetically. It is therefore not homogeneous, as would be the case if the crystal were a uniform solid solution throughout. Analyses and magnetic measurements on a cross-section of an Elba crystal showed that the magnetic susceptibility and percentage of FeO vary, not irregularly, but continuously, being highest at the base and lowest at the free-growing tip of the crystal. The crystal is therefore zoned with respect to its FeO content.

Since  $\text{Fe}_3\text{O}_4$  goes into solid solution in  $\text{Fe}_2\text{O}_3$ , forming a single solid phase of varying composition and properties, a zonal distribution of FeO is to be expected in an oxide of iron depositing from a vapor or solution. The occurrence of such zonal growth indicates continuously changing conditions of temperature, pressure and concentration during the formation of the crystals. Several ore deposits of contact-metamorphic origin show a zonal distribution of ferrous iron, probably arising from the same causes as the zoning of the single crystals.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## A Standard Screen Scale for Testing Sieves\*

(St. Louis Meeting, October, 1917)

SINCE the adoption by the U. S. Bureau of Standards several years ago of specifications for standard 100- and 200-mesh sieves, frequent requests have been received that this Bureau test and certify sieves of other sizes than these. With a view to the adoption of a series of standard testing sieves which might be of use to all industries making fineness tests, this Bureau for two years has been studying the question of such a standard screen scale. Various scales that have been proposed were considered, and information was sought of representative firms in the various industries interested as to their requirements. Manufacturers of sieves have also been consulted as to the desirability of different screen scales and the practicability of their manufacture. As a result of this study of the question, a conference was called at the Bureau of Standards, Apr. 20, 1916, of representatives of various committees of the American Society for Testing Materials, American Society of Civil Engineers, American Institute of Mining Engineers, American Foundrymen's Association, [Mining and Metallurgical Society of America, American Water Works Association, American Institute of Metals and the American Spice Trade Association; also representatives of the Committee of Revision of the U. S. Pharmacopœia, the U. S. Geological Survey, the U. S. Bureau of Mines, the U. S. Office of Public Roads and Rural Engineering, the U. S. Office of Grain Standardization and the U. S. Bureau of Standards; also representatives of a number of private firms engaged in industries in which sieves are used, such as the glass, the drug milling, the abrasive, the asphalt, the mining, the spice, the chemical, and the graphite industries; also representatives of the firms in this country manufacturing wire cloth and sieves.

This Conference, after considering the various screen scales either proposed or now in use, adopted as a Standard Screen Scale that given in Table 1, and recommends that it be adopted generally by scientific,

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\* Adopted by a conference of representatives of various scientific and technical societies, government bureaus, and private firms, held at the Bureau of Standards, and recommended for general adoption in the interests of securing uniformity of usage.

TABLE 1.—*Standard Screen Scale*

Based on a 1-mm. opening sieve with the square root of 2, or 1.4142, as the ratio of the openings of successive sieves coarser than 1 mm., and the fourth root of 2, or 1.1892, as the ratio of the openings of successive sieves finer than 1 mm.

|                 | Opening | Mesh | Wire<br>Diam. | Ratio                       | Tolerances |         |
|-----------------|---------|------|---------------|-----------------------------|------------|---------|
|                 |         |      |               | Wire<br>Diam. to<br>Opening | Mesh       | Diam.   |
| 8-mm. Sieve:    |         |      |               |                             |            |         |
| Metric.....     | 8.00    | 1.0  | 2.00          | 0.25                        | ±0.01      | ±0.08   |
| Customary.....  | 0.315   | 2.54 | 0.079         | ....                        | ±0.03      | ±0.003  |
| 5.66-mm. Sieve: |         |      |               |                             |            |         |
| Metric.....     | 5.66    | 1.4  | 1.48          | 0.26                        | ±0.01      | ±0.08   |
| Customary.....  | 0.223   | 3.56 | 0.058         | ....                        | ±0.03      | ±0.003  |
| 4-mm. Sieve:    |         |      |               |                             |            |         |
| Metric.....     | 4.00    | 2.0  | 1.00          | 0.25                        | ±0.02      | ±0.05   |
| Customary.....  | 0.157   | 5.1  | 0.039         | ....                        | ±0.05      | ±0.002  |
| 2.83-mm. Sieve: |         |      |               |                             |            |         |
| Metric.....     | 2.83    | 2.75 | 0.81          | 0.29                        | ±0.02      | ±0.05   |
| Customary.....  | 0.111   | 7.0  | 0.032         | ....                        | ±0.05      | ±0.002  |
| 2-mm. Sieve:    |         |      |               |                             |            |         |
| Metric.....     | 2.00    | 3.9  | 0.56          | 0.28                        | ±0.04      | ±0.05   |
| Customary.....  | 0.079   | 9.9  | 0.022         | ....                        | ±0.1       | ±0.002  |
| 1.41-mm. Sieve: |         |      |               |                             |            |         |
| Metric.....     | 1.41    | 5.0  | 0.59          | 0.42                        | ±0.08      | ±0.025  |
| Customary.....  | 0.0555  | 12.7 | 0.0232        | ....                        | ±0.2       | ±0.0010 |
| 1-mm. Sieve:    |         |      |               |                             |            |         |
| Metric.....     | 1.00    | 7.0  | 0.43          | 0.43                        | ±0.15      | ±0.020  |
| Customary.....  | 0.0394  | 17.8 | 0.0169        | ....                        | ±0.4       | ±0.0008 |
| 0.85-mm. Sieve: |         |      |               |                             |            |         |
| Metric.....     | 0.85    | 8.0  | 0.40          | 0.47                        | ±0.2       | ±0.015  |
| Customary.....  | 0.0335  | 20.3 | 0.0157        | ....                        | ±0.5       | ±0.0006 |
| 0.71-mm. Sieve: |         |      |               |                             |            |         |
| Metric.....     | 0.71    | 9.0  | 0.40          | 0.56                        | ±0.3       | ±0.012  |
| Customary.....  | 0.0280  | 22.9 | 0.0157        | ....                        | ±0.75      | ±0.0005 |
| 0.59-mm. Sieve: |         |      |               |                             |            |         |
| Metric.....     | 0.59    | 10.0 | 0.41          | 0.69                        | ±0.4       | ±0.012  |
| Customary.....  | 0.0232  | 25.4 | 0.0161        | ....                        | ±1.0       | ±0.0005 |
| 0.5-mm. Sieve:  |         |      |               |                             |            |         |
| Metric.....     | 0.50    | 12.0 | 0.33          | 0.66                        | ±0.4       | ±0.012  |
| Customary.....  | 0.0197  | 30.5 | 0.0130        | ....                        | ±1.0       | ±0.0005 |
| 0.42-mm. Sieve: |         |      |               |                             |            |         |
| Metric.....     | 0.42    | 14.0 | 0.29          | 0.69                        | ±0.6       | ±0.010  |
| Customary.....  | 0.0165  | 35.6 | 0.0114        | ....                        | ±1.5       | ±0.0004 |
| 0.36-mm. Sieve: |         |      |               |                             |            |         |
| Metric.....     | 0.36    | 16.0 | 0.26          | 0.72                        | ±0.6       | ±0.010  |
| Customary.....  | 0.0142  | 40.6 | 0.0102        | ....                        | ±1.5       | ±0.0004 |



TABLE 1.—*Standard Screen Scale.*—(Continued)

|                  | Opening | Mesh  | Wire<br>Diam. | Ratio                       | Tolerances |          |
|------------------|---------|-------|---------------|-----------------------------|------------|----------|
|                  |         |       |               | Wire<br>Diam. to<br>Opening | Mesh       | Diam.    |
| 0.29-mm. Sieve:  |         |       |               |                             |            |          |
| Metric.....      | 0.29    | 20.0  | 0.21          | 0.72                        | ±0.8       | ±0.010   |
| Customary.....   | 0.0114  | 50.8  | 0.0083        | ....                        | ±2.0       | ±0.0004  |
| 0.25-mm. Sieve:  |         |       |               |                             |            |          |
| Metric.....      | 0.25    | 23.0  | 0.185         | 0.74                        | ±1.0       | ±0.008   |
| Customary.....   | 0.0098  | 58.4  | 0.0073        | ....                        | ±3.0       | ±0.0003  |
| 0.21-mm. Sieve:  |         |       |               |                             |            |          |
| Metric.....      | 0.21    | 27.0  | 0.16          | 0.76                        | ±1.0       | ±0.008   |
| Customary.....   | 0.0083  | 68.6  | 0.0063        | ....                        | ±3.0       | ±0.0003  |
| 0.17-mm. Sieve:  |         |       |               |                             |            |          |
| Metric.....      | 0.17    | 31.0  | 0.15          | 0.88                        | ±1.0       | ±0.008   |
| Customary.....   | 0.0067  | 78.7  | 0.0059        | ....                        | ±3.0       | ±0.0003  |
| 0.14-mm. Sieve:  |         |       |               |                             |            |          |
| Metric.....      | 0.14    | 39.0  | 0.116         | 0.83                        | ±1.0       | ±0.008   |
| Customary.....   | 0.0055  | 99.1  | 0.0046        | ....                        | ±3.0       | ±0.0003  |
| 0.125-mm. Sieve: |         |       |               |                             |            |          |
| Metric.....      | 0.125   | 47.0  | 0.089         | 0.71                        | ±1.5       | ±0.008   |
| Customary.....   | 0.0049  | 119.4 | 0.0035        | ....                        | ±4.0       | ±0.0003  |
| 0.105-mm. Sieve: |         |       |               |                             |            |          |
| Metric.....      | 0.105   | 59.0  | 0.064         | 0.61                        | ±2.0       | ±0.008   |
| Customary.....   | 0.0041  | 149.9 | 0.0025        | ....                        | ±5.0       | ±0.0003  |
| 0.088-mm. Sieve: |         |       |               |                             |            |          |
| Metric.....      | 0.088   | 67.0  | 0.061         | 0.69                        | ±2.5       | ±0.005   |
| Customary.....   | 0.0035  | 170.2 | 0.0024        | ....                        | ±6.0       | ±0.0002  |
| 0.074-mm. Sieve: |         |       |               |                             |            |          |
| Metric.....      | 0.074   | 79.0  | 0.053         | 0.72                        | ±3.0       | ±0.005   |
| Customary.....   | 0.0029  | 200.7 | 0.0021        | ....                        | ±8.0       | ±0.0002  |
| 0.062-mm. Sieve: |         |       |               |                             |            |          |
| Metric.....      | 0.062   | 98.0  | 0.040         | 0.65                        | ±3.5       | ±0.005   |
| Customary.....   | 0.0024  | 248.9 | 0.0016        | ....                        | ±9.0       | ±0.0002  |
| 0.052-mm. Sieve: |         |       |               |                             |            |          |
| Metric.....      | 0.052   | 110.0 | 0.039         | 0.72                        | ±4.0       | ±0.004   |
| Customary.....   | 0.0021  | 279.4 | 0.0015        | ....                        | ±10.0      | ±0.00015 |
| 0.044-mm. Sieve: |         |       |               |                             |            |          |
| Metric.....      | 0.044   | 127.0 | 0.035         | 0.80                        | ±5.0       | ±0.004   |
| Customary.....   | 0.0017  | 323.0 | 0.0014        | ....                        | ±12.0      | ±0.00015 |

technical, and engineering societies and committees, and by branches of National, State, and Municipal Governments as a part of their specifications for materials and methods of test; also that it be used by private firms who have need of standard sieves.

This screen scale is essentially metric. The sieve having an opening of 1 mm. is the basic one, and the sieves above and below this in the series are related to it by using in general the square root of 2, or 1.4142, or

the fourth root of 2, or 1.1892, as the ratio of the width of one opening to the next smaller opening. The first ratio is used for openings between 1 mm. and 8 mm. while the fourth root of 2 is used as the ratio for openings below 1 mm. to give more sieves in that part of the scale. The series has been made large enough, it is hoped, to meet the needs of all industries. Some industries may have occasion to use all the sieves in a certain section of the series and none of the others, while in other industries it may be desirable to use only certain sieves selected from the whole range of the series. In making such selections it is recommended that this be done on some systematic plan as, for example, the selection of every other sieve or of every fourth one in the series below 1 mm. opening and every other sieve above 1 mm., in which case the ratio of each opening to the next smaller one would be as 2 to 1.

Because of the wide range of openings in sieves now manufactured which are possible with a given number of meshes of wire per unit length by the use of wires of different diameters, and the consequent confusion and uncertainty which arises in designating sieves by the number of meshes per unit length, the sieves of this series have been designated by the width of the opening in millimeters as, for example, a 1.41-mm. sieve, or a 0.36-mm. sieve. It is urgently recommended that all users of sieves in the future designate these standards sieves in this way and that the manufacturers mark and list the sieves in this manner rather than by the meshes per inch.

In the designation and certification of the sieves the metric units will be used by the Bureau of Standards. In Table 1, however, are also given the equivalents of these metric quantities in inches in order that the series may be more readily related to work previously done. It is, of course, immaterial whether units of the metric system, or of the customary system, or of any other system, are used in the manufacture of the sieves, provided they are within the tolerances.

To meet the need for sieves of this series at the present time, a temporary provision has been made in the specifications for the acceptance of sieves of slightly different mesh and wire diameter than that called for in the screen scale, provided the resultant opening is the same as the nominal opening within a small range. This will make possible the use of a number of sieves now on the market in which the ratio of wire diameter to opening is only slightly different from that of the screen scale. This provision will be withdrawn when conditions are such that the manufacturers of sieves can furnish sieves made more exactly in accordance with the specifications.

The Bureau of Standards hereby announces that it will test sieves of this series to determine whether they conform to specifications given below. This test will consist of the examination of the mesh of both the warp and shoot wires of the cloth to ascertain whether it comes within

the tolerances allowed; also measurements of the diameter of wires in each direction to determine the average diameter, and a measurement of any large openings to ascertain whether they exceed the limits given in these specifications; also an examination of the sieve to discover any imperfections of the sieve which may seriously affect its sieving value. Sieves which pass the specifications will be stamped with the seal of this Bureau and will be given an identification number. A certificate will be furnished for each sieve that passes the requirements.

For sieves which fail to meet the specifications, reports will be rendered showing wherein the sieve was not up to the standard.

A fee of \$2 per sieve will be charged for the test of the sieves when submitted singly. For from 2 to 9 sieves submitted at one time the fee will be \$1.50 per sieve. For lots of 10 or more the fee will be \$1 per sieve. Only half of the above fees will be charged for such sieves as may be rejected for exceeding the tolerances of mesh, in which case the wire diameter will not be measured.

In Table 1, in the first column headed "Openings," is given the width of the opening (on the first line in millimeters, on the second line in inches), for each sieve. In the second column, headed "Mesh," on the first line is given the number of meshes per linear centimeter, and on the second line the equivalent number of meshes per linear inch. In the third column, headed "Wire Diam.," is given on the first line the diameter of wire in millimeters, and on the second line its equivalent in inches. In the fourth column, headed "Ratio Wire Diam. to Opening," is given the ratio of the wire diameter to the width of the opening between wires. In the fifth and sixth columns, headed "Tolerances," are given the tolerances for these sieves mentioned in the specifications below. These tolerances will, for testing purposes, be used essentially in the metric dimension, but on the second line in each case is given the equivalent in inches in order that the tolerances may be compared readily with those in previous use. The tolerance in the fifth column is that for the meshes per centimeter and per inch, respectively, and in the sixth column the tolerances for wire diameter in millimeters and inches, on the first and second lines respectively.

In Table 2 is given a list showing the dimensions of sieves now on the market which would most nearly meet the specifications and tolerances of the Standard Screen Scale. The headings of the columns of this table are self-explanatory. Where the dimensions of more than one sieve are shown for a given sieve of the screen scale, one set of dimensions is that of one manufacturer, and the other that of another. In some cases the third set, if one is given, is made by two or more manufacturers.

In Table 3 is given a list of sieves between 1-mm. opening and 8-mm. opening which would be interpolated in the series of the Standard Screen Scale if the fourth root of 2 or 1.1892, were used as the ratio of successive

TABLE 2

Sieves now on the market which would most nearly meet the tolerances of the Standard Screen Scale.

|                    | Opening<br>in Mm. | Opening<br>in Inches | Meshee<br>per Inch | Wire Diam.<br>in Inches | Wire Diam.<br>in Mm. |
|--------------------|-------------------|----------------------|--------------------|-------------------------|----------------------|
| 8-mm. Sieve.....   | 8.13              | 0.320                | 2.5                | 0.080                   | 2.03                 |
|                    | 8.05              | 0.317                | 2.5                | 0.083                   | 2.11                 |
| 5.66-mm. Sieve.... | 5.66              | 0.223                | 3.5                | 0.063                   | 1.60                 |
|                    | 5.61              | 0.221                | 3.5                | 0.065                   | 1.65                 |
| 4-mm. Sieve.....   | 4.04              | 0.159                | 5.0                | 0.041                   | 1.04                 |
|                    | 4.06              | 0.160                | 5.0                | 0.040                   | 1.02                 |
| 2.83-mm. Sieve.... | 2.82              | 0.111                | 7.0                | 0.032                   | 0.81                 |
|                    | 2.82              | 0.111                | 7.0                | 0.0315                  | 0.80                 |
| 2-mm. Sieve.....   | 1.96              | 0.077                | 10.0               | 0.023                   | 0.58                 |
|                    | 2.03              | 0.080                | 10.0               | 0.020                   | 0.51                 |
|                    | 2.03              | 0.080                | 10.0               | 0.0205                  | 0.52                 |
| 1.41-mm. Sieve.... | 1.40              | 0.055                | 12.0               | 0.028                   | 0.71                 |
|                    | 1.42              | 0.056                | 12.0               | 0.027                   | 0.69                 |
| 1-mm. Sieve.....   | 1.01              | 0.0396               | 18.0               | 0.016                   | 0.41                 |
|                    | 0.99              | 0.0391               | 18.0               | 0.0165                  | 0.42                 |
| 0.85-mm. Sieve.... | 0.85              | 0.0335               | 20.0               | 0.0165                  | 0.42                 |
|                    | 0.86              | 0.0340               | 20.0               | 0.016                   | 0.41                 |
| 0.71-mm. Sieve.... | 0.72              | 0.0285               | 22.0               | 0.017                   | 0.43                 |
|                    | 0.70              | 0.0275               | 22.0               | 0.018                   | 0.46                 |
|                    | 0.74              | 0.0290               | 22.0               | 0.0165                  | 0.42                 |
| 0.59-mm. Sieve ... | 0.58              | 0.0230               | 26.0               | 0.0155                  | 0.39                 |
|                    | 0.60              | 0.0235               | 26.0               | 0.015                   | 0.38                 |
| 0.5-mm. Sieve..... | 0.50              | 0.0198               | 30.0               | 0.0135                  | 0.34                 |
|                    | 0.50              | 0.0196               | 30.0               | 0.01375                 | 0.35                 |
| 0.42-mm. Sieve.... | 0.42              | 0.0166               | 35.0               | 0.012                   | 0.30                 |
|                    | 0.42              | 0.0164               | 35.0               | 0.01225                 | 0.31                 |
| 0.36-mm. Sieve.... | 0.36              | 0.0140               | 40.0               | 0.011                   | 0.28                 |
|                    | 0.35              | 0.01375              | 40.0               | 0.01125                 | 0.29                 |
|                    | 0.37              | 0.01475              | 40.0               | 0.01025                 | 0.26                 |
| 0.29-mm. Sieve.... | 0.28              | 0.0110               | 50.0               | 0.009                   | 0.23                 |
| 0.25-mm. Sieve.... | 0.25              | 0.0097               | 60.0               | 0.007                   | 0.18                 |
|                    | 0.26              | 0.0102               | 60.0               | 0.0065                  | 0.17                 |
|                    | 0.23              | 0.0092               | 60.0               | 0.0075                  | 0.19                 |
| 0.21-mm. Sieve.... | 0.19              | 0.0073               | 70.0               | 0.007                   | 0.18                 |
|                    | 0.20              | 0.0078               | 70.0               | 0.0065                  | 0.17                 |
| 0.17-mm. Sieve.... | 0.17              | 0.0068               | 80.0               | 0.00575                 | 0.15                 |
| 0.14-mm. Sieve.... | 0.14              | 0.0055               | 100.0              | 0.0045                  | 0.114                |
| 0.125-mm. Sieve... | 0.117             | 0.0046               | 120.0              | 0.0037                  | 0.094                |
|                    | 0.119             | 0.0047               | 120.0              | 0.0036                  | 0.091                |
| 0.105-mm. Sieve... | 0.104             | 0.0041               | 150.0              | 0.0026                  | 0.066                |
|                    | 0.094             | 0.0037               | 150.0              | 0.0030                  | 0.076                |
| 0.088-mm. Sieve... | 0.089             | 0.0035               | 170.0              | 0.0024                  | 0.061                |
|                    | 0.084             | 0.0033               | 170.0              | 0.0026                  | 0.066                |
| 0.074-mm. Sieve... | 0.074             | 0.0029               | 200.0              | 0.0021                  | 0.053                |
| 0.062-mm. Sieve... | 0.061             | 0.0024               | 250.0              | 0.0016                  | 0.041                |
|                    | 0.058             | 0.0023               | 250.0              | 0.0017                  | 0.043                |
| 0.052-mm. Sieve... | 0.051             | 0.0020               | 280.0              | 0.0016                  | 0.041                |
| 0.044-mm. Sieve... | 0.041             | 0.0016               | 325.0              | 0.0015                  | 0.038                |
|                    | 0.041             | 0.0016               | 330.0              | 0.0014                  | 0.036                |

sieves throughout the series. Suitable meshes and wire diameters to give these openings are also given, together with the tolerances under which such sieves would be tested if used. These sieves have not been included in the Standard Screen Scale, as it is believed to be unnecessary to have so many sieves in this part of the scale. This list is given separately, however, in case any organization in selecting sieves systematically from the Standard Screen Scale in the series of openings less than 1 mm. finds it desirable to use any of these interpolated sieves above 1 mm. in carrying out a systematic plan of selection of sieves. In case any organization or firm should adopt any of these six sieves under such circumstances, the Bureau of Standards will test and certify them in accordance with the dimensions given herewith.

TABLE 3

Additional sieves which would be interpolated between the 8-mm. opening and the 1-mm. opening of the Standard Screen Scale by the use of the fourth root of 2, or 1.1892, as the ratio of the successive sieve-openings.

|                 | Opening | Mesh | Wire<br>Diam. | Ratio<br>Wire<br>Diam. to<br>Opening | Tolerances |         |
|-----------------|---------|------|---------------|--------------------------------------|------------|---------|
|                 |         |      |               |                                      | Mesh       | Diam.   |
| 6.72-mm. Sieve: |         |      |               |                                      |            |         |
| Metric.....     | 6.72    | 1.2  | 1.61          | 0.24                                 | ±0.01      | ±0.08   |
| Customary.....  | 0.265   | 3.05 | 0.063         | ....                                 | ±0.03      | ±0.003  |
| 4.76-mm. Sieve: |         |      |               |                                      |            |         |
| Metric.....     | 4.76    | 1.6  | 1.49          | 0.31                                 | ±0.02      | ±0.05   |
| Customary.....  | 0.187   | 4.1  | 0.059         | ....                                 | ±0.05      | ±0.002  |
| 3.36-mm. Sieve: |         |      |               |                                      |            |         |
| Metric.....     | 3.36    | 2.4  | 0.81          | 0.24                                 | ±0.02      | ±0.05   |
| Customary.....  | 0.132   | 6.1  | 0.032         | ....                                 | ±0.05      | ±0.002  |
| 2.38-mm. Sieve: |         |      |               |                                      |            |         |
| Metric.....     | 2.38    | 3.15 | 0.79          | 0.33                                 | ±0.04      | ±0.05   |
| Customary.....  | 0.094   | 8.0  | 0.031         | ....                                 | ±0.1       | ±0.002  |
| 1.68-mm. Sieve: |         |      |               |                                      |            |         |
| Metric.....     | 1.68    | 4.0  | 0.82          | 0.49                                 | ±0.04      | ±0.025  |
| Customary.....  | 0.067   | 10.2 | 0.032         | ....                                 | ±0.1       | ±0.0010 |
| 1.19-mm. Sieve: |         |      |               |                                      |            |         |
| Metric.....     | 1.19    | 6.0  | 0.48          | 0.40                                 | ±0.1       | ±0.020  |
| Customary.....  | 0.0468  | 15.2 | 0.0189        | ....                                 | ±0.25      | ±0.0008 |

In Table 4 is given an extension of the metric series beyond the 8-mm. opening using the square root of 2, or 1.4142, as the ratio of successive openings. A suggested diameter of wire for use in making such sieves is also given. No tolerances for these sieves are given, however, as it is not proposed to test such sieves at the Bureau of Standards, since the user of the sieves could ordinarily make such tests of them as are necessary with sufficient accuracy. Such sieves would generally be made

up into sieves of larger diameter than those of the Standard Screen Scale and would usually be made of iron or steel wire.

TABLE 4

Showing the sieves which an extension of the Standard Screen Scale from an 8-mm. opening to a 128-mm. opening would comprise, using the square root of 2, or 1.4142, as the ratio of successive openings.

|                        | Opening | Wire Diameter |
|------------------------|---------|---------------|
| <b>128-mm. Sieve:</b>  |         |               |
| Metric.....            | 128.0   | 9.5           |
| Customary.....         | 5.04    | 0.375         |
| <b>90.5-mm. Sieve:</b> |         |               |
| Metric.....            | 90.5    | 9.5           |
| Customary.....         | 3.56    | 0.375         |
| <b>64-mm. Sieve:</b>   |         |               |
| Metric.....            | 64.0    | 6.4           |
| Customary.....         | 2.52    | 0.25          |
| <b>45.3-mm. Sieve:</b> |         |               |
| Metric.....            | 45.3    | 5.26          |
| Customary.....         | 1.78    | 0.207         |
| <b>32-mm. Sieve:</b>   |         |               |
| Metric.....            | 32.0    | 4.85          |
| Customary.....         | 1.26    | 0.192         |
| <b>22.6-mm. Sieve:</b> |         |               |
| Metric.....            | 22.6    | 4.11          |
| Customary.....         | 0.891   | 0.162         |
| <b>16-mm. Sieve:</b>   |         |               |
| Metric.....            | 16.0    | 3.05          |
| Customary.....         | 0.630   | 0.120         |
| <b>11.3-mm. Sieve:</b> |         |               |
| Metric.....            | 11.3    | 2.67          |
| Customary.....         | 0.445   | 0.105         |

In the specifications given below the diameter or other dimensions of the sieve frames are not given, with the idea that any organization or firm in adopting these specifications will decide upon the size of sieve frame that best meets its needs. For purposes of uniformity and interchangeability of sieves, pans, and covers, it is recommended that sieves be purchased in diameters of either 20 cm., 15 cm., or 10 cm. (7.87 in., 5.91 in., or 3.94 in.). These are the outside diameters of the bottom of the sieve or the inside diameter of the top of the sieve.

#### SPECIFICATIONS FOR SIEVES OF THE STANDARD SCREEN SCALE

Wire cloth for standard sieves shall be woven (not twilled, except that the cloth of 0.062-mm., the 0.052-mm., and the 0.044-mm. sieve,

may be twilled until further notice), from brass, bronze, or other suitable wire and mounted on the frames without distortion. To prevent the material being sieved from catching in the joint between the cloth and the frame, the joint shall be smoothly filled with solder, or so made that the material will not catch.

The number of wires per centimeter of the cloth of any given sieve of the Standard Screen Scale shall be that shown in the accompanying Table 1 in the second column, headed "Mesh," and the number of wires in any whole centimeter shall not differ from this amount by more than the tolerance given in the 5th column, that headed "Mesh" under the heading "Tolerances." No opening between adjacent parallel wires shall be greater than the nominal width of opening for that sieve by more than the following amounts:

10 per cent. of the nominal width of opening for the 8-mm. to 1-mm. sieve, inclusive.

25 per cent. of the nominal width of opening for the 0.85-mm. to the 0.29-mm. sieve, inclusive.

40 per cent. of the nominal width of opening for the 0.25-mm. to the 0.125-mm. sieve, inclusive.

60 per cent. of the nominal width of opening for the 0.105-mm. to the 0.044-mm. sieve, inclusive.

The diameters of the wires of the cloth of any given sieve shall be that shown in the third column of Table 1 headed "Wire Diam." and the average diameter of the wires in either direction shall not differ from the specified diameter by more than the tolerance given in the last column of Table 1, that under "Tolerances" headed "Diam."

The Bureau of Standards also reserves the right to reject sieves for obvious imperfections in the sieve cloth or its mounting, as, for example, punctured, loose or wavy cloth, imperfections in soldering, etc.

Until further notice, to permit the use of sieves now on the market which have slightly different mesh and wire diameters from that specified above, sieves will be certified as satisfactory if the measurements of mesh and wire diameters show the resulting average width of opening to be within 4 per cent. of the nominal opening of a given sieve, and the ratio of wire diameter to opening of the sieve in question is within 0.03 of that given in Table 1 in the column headed "Ratio Wire Diam. to Opening" for the 8-mm. to the 2-mm. sieves, inclusive, and within 0.06 of the ratio given for sieves of smaller openings than 2 mm.





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## A Feasible Plan for Gaging Individual Wells

BY ROSWELL H. JOHNSON,\* B. S., M. S., AND W. E. BERNARD, PITTSBURGH, PA.

(St. Louis Meeting, October, 1917)

To know the rate of decline of oil wells is very important, yet ordinarily we are prevented from getting this rate because the oil from several wells is put into one or a few tanks as soon as the wells have been connected to the power. The object of doing this is economy of tanks, pipes, labor and accounting.

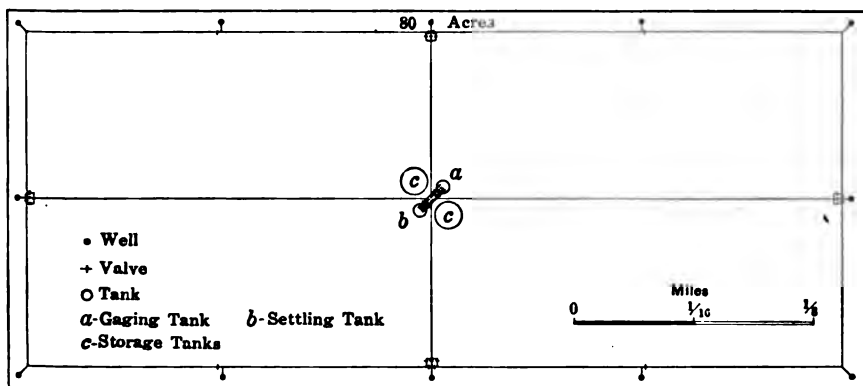


FIG. 1.

But we are making a great sacrifice for this small economy. The advantages of the individual records are that we would know better:

1. When the well has ceased to be profitable.
2. When a well has gotten into bad condition.
3. The real record of the yield per acre and the rate of decline of the wells. The lease total is too mixed as to age of the wells for existing records to have much significance. To obtain such individual well records and more accurate logs are the great desiderata of satisfactory management of oil properties.
4. Which wells show the slowest decline—information which is valuable in locating new wells or purchasing adjoining leases.

The truth of this statement is evident, when we reflect upon the large number of purchases of producing property. Intelligent appraisal is of the utmost advantage and it is impossible without individual well-pro-

\* Professor of Oil and Gas Production, University of Pittsburgh.

duction figures. Such figures are valuable not only for understanding the lease in question, but for making comparisons.

In contemplating the wide variation in the estimation of values, one is impressed with the importance of ascertaining the data that are so fundamental to appraising.

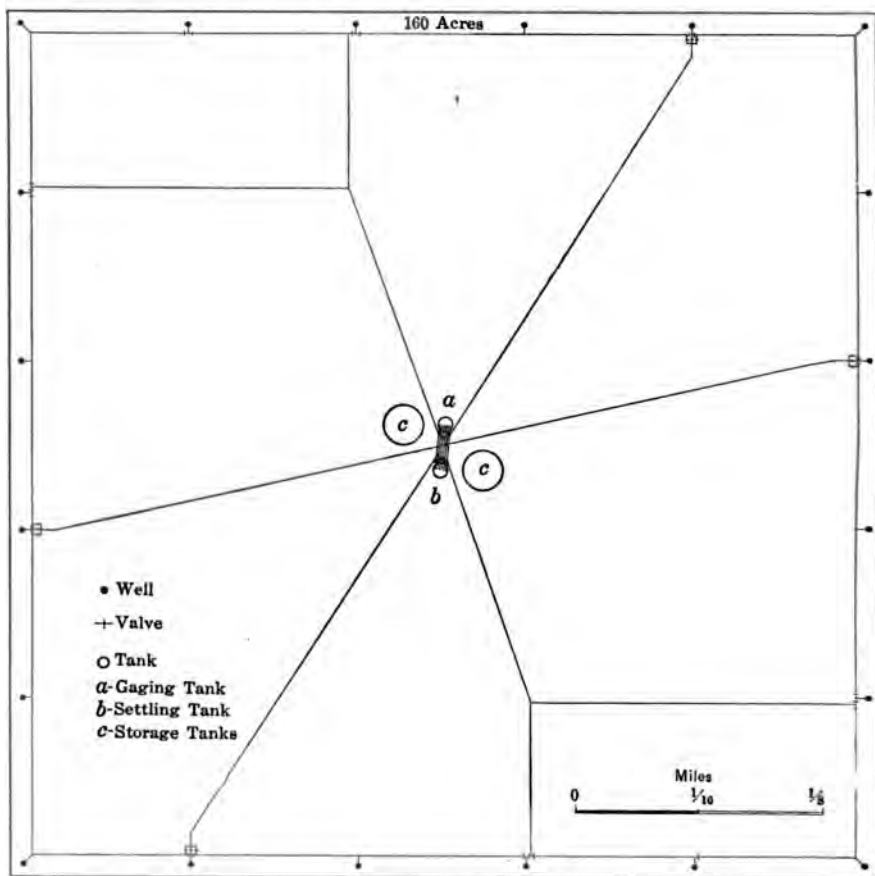


FIG. 2.

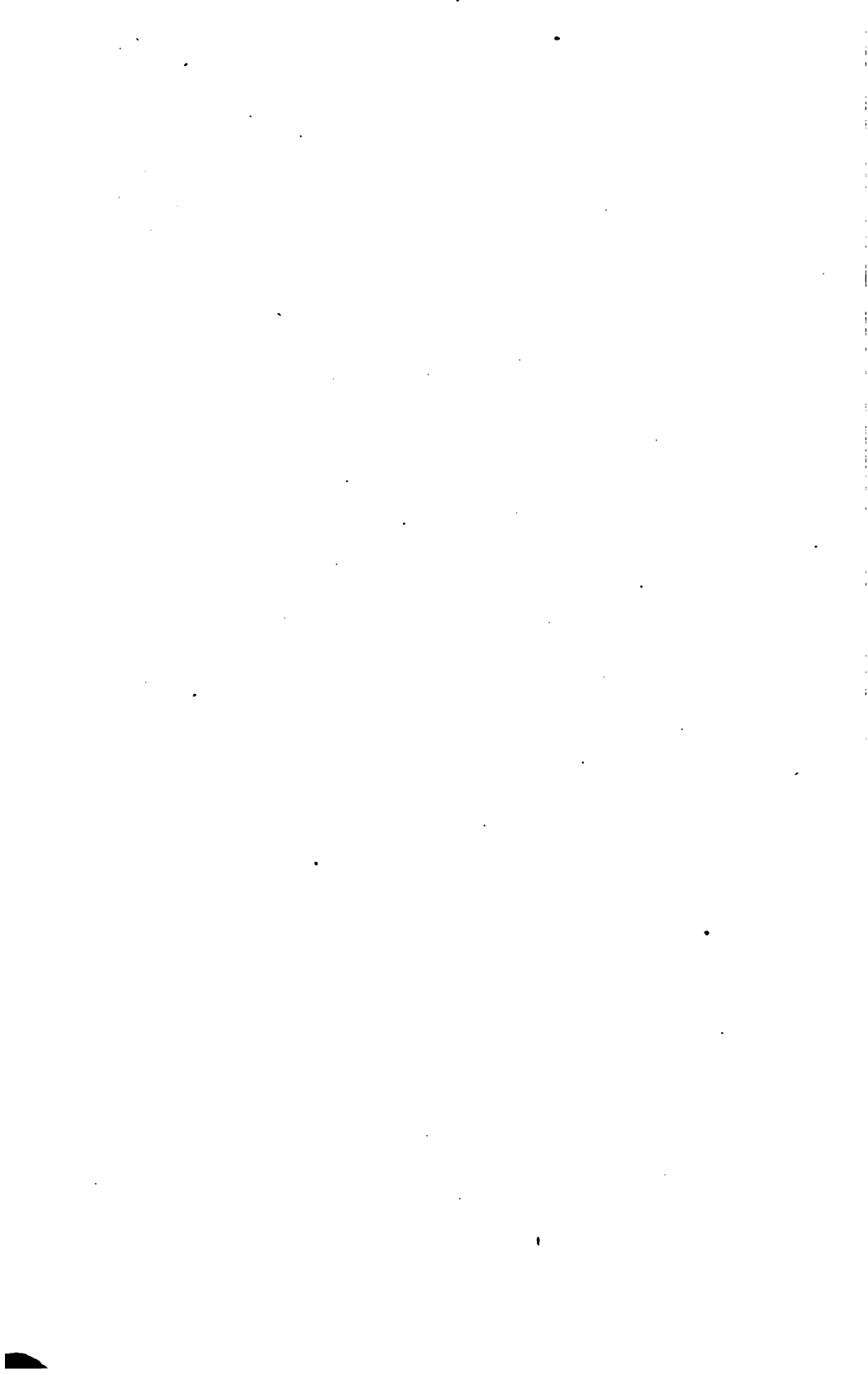
Many who would concede the general argument so far, still consider individual gaging too expensive. To reduce this expense we have devised a plan, whereby an individual gaging is taken of each well in rotation. Each well produces into the special measuring tank one day and then into the general receiving tank until its turn comes in rotation for another individual gage. Such records will be frequent enough to give the advantages listed.

The pipe required is far less than would be imagined owing to a scheme of valve manipulation we have devised. Figs. 1 and 2 show the arrange-

ments on leases of 80 and 160 acres respectively. On 10- and 40-acre leases separate radial lines are used.

The cost of the additional materials required, exclusive of the gaging tank, teaming and labor, is at present prices \$50, \$56 and \$20 per well for a 160-, 80- and 40-acre lease, respectively, as compared with the most economical possible arrangement.

Should this expense for the 80- and 160-acre leases be thought prohibitive for routine use, it is recommended for the 10- and 40- acre leases as in these cases the additional expense is not so great per well. A compromise plan in larger leases would be individual tanks for certain wells the performance of which would have especial significance and separate tanking from shallow and deep sands.



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## The Replacement of Sulphides by Quartz

BY H. N. WOLCOTT,\* B. S., A. M., BISBEE, ARIZ.

(St. Louis Meeting, October, 1917)

AMONG the many cases of replacement of one mineral by another, that of quartz or silicates by pyrite, or even other sulphides, is not uncommon, but the reverse of this process does not appear to have been recorded, even if it has been observed. It, therefore, seemed to the writer that the case described below was worthy of record.

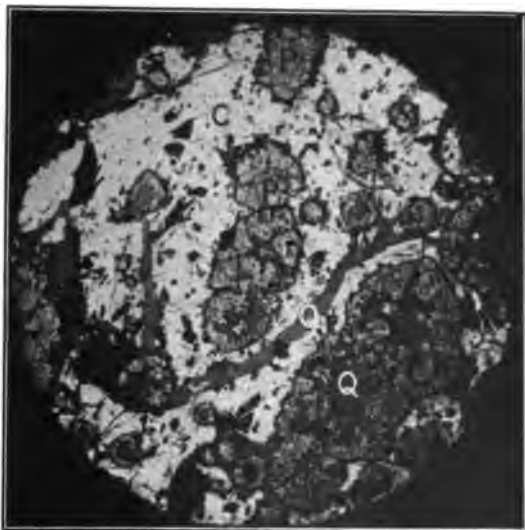


FIG. 1.—GROUND MASS OF CHALCOPYRITE (C) CONTAINING ROUNDED GRAINS OF PYRITE (P) SHOWING DIFFERENT DEGREES OF REPLACEMENT BY QUARTZ (Q), AND ALSO QUARTZ VEINLETS CUTTING THROUGH THE PYRITE. (PHOTOGRAPHED BY REFLECTED LIGHT.)

The specimens described in this paper were obtained by R. B. Morton of Idaho Springs, Colo., from the Old Town Mine, in Russell Gulch, and are in the collection of Cornell University, where the writer studied them.

In the rough, the specimens appeared to consist largely of chalcopyrite, with some grains of tetrahedrite. The polished surface of some of

\* Geologist, Copper Queen Consolidated Mining Co.

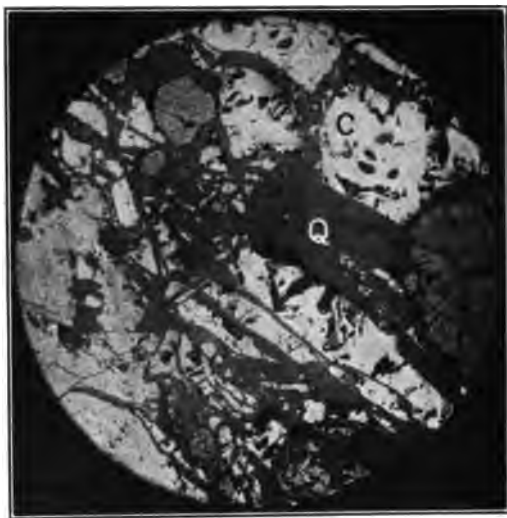


FIG. 2.—BRECCIATED ORE SHOWING, CHALCOPYRITE (C), PYRITE (P) AND QUARTZ (Q). (PHOTOGRAPHED BY REFLECTED LIGHT.)

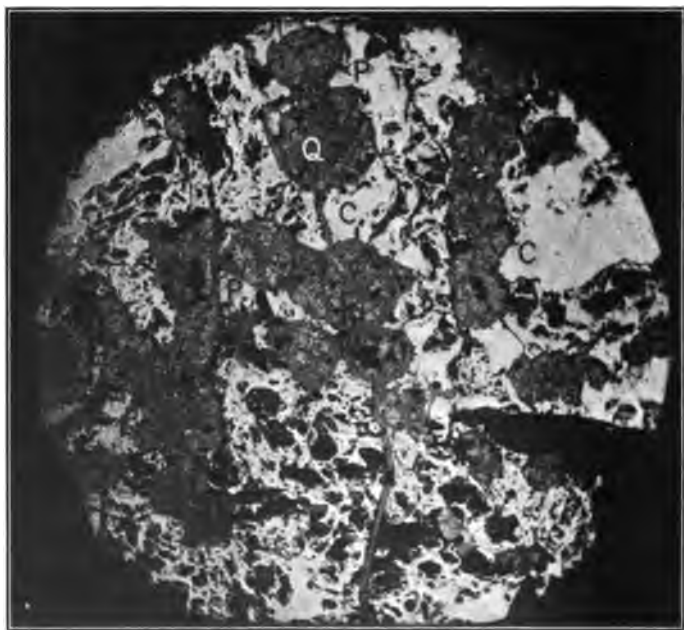


FIG. 3.—CHALCOPYRITE (C), CONTAINING PYRITE (P), AND VEINLETS OF QUARTZ (Q). THE LAST NAMED NOT ONLY CUTS THE CHALCOPYRITE, BUT REPLACES IT AROUND SOME OF THE PARTLY REPLACED PYRITE GRAINS, AS SHOWN BY CRYSTAL OUTLINES OF QUARTZ (METACRYSTS.) BLACK SPOTS ARE DEPRESSIONS IN SURFACE. (PHOTOGRAPHED BY REFLECTED LIGHT.)

the lumps, however, showed numerous rounded grains of pyrite, and minute fractures.

These smooth surfaces studied by reflected light under the microscope exhibited most interesting relationships. There, it was seen that while chalcopyrite and tetrahedrite were probably contemporaneous, numerous grains and rosettes of pyrite of varying size were scattered through the chalcopyrite. A few were seen in the tetrahedrite. As the pyrite is completely surrounded by the chalcopyrite (and also the tetrahedrite) it may be of earlier age.

The pyrite rosettes and grains consist in some instances entirely of that mineral. In other cases, and by far the most numerous, the rosettes show an irregular central mass of quartz (Fig. 1), of varying size, and often connecting with the exterior by one or more veinlets of quartz (Fig. 2).

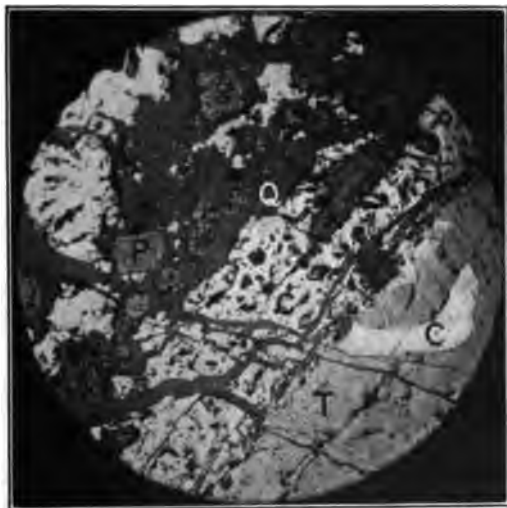


FIG. 4.—CHALCOPYRITE (C), AND TETRAHEDRITE (T), CUT BY QUARTZ VEINS PYRITE (P), IN PART REPLACED.

Some show a little quartz on the outside of the pyrite grain, and in one case at least (Fig. 3), the quartz surrounding the pyrite showed the crystal outlines of quartz.

The quartz, in addition to its occurrence in the pyrite grains, fills numerous narrow fractures in the chalcopyrite and tetrahedrite, indeed in some parts of the specimen, the chalcopyrite and tetrahedrite have been crushed to a mass of angular fragments (Fig. 2) with quartz filling the crevices so formed. One other point to be noted is that the quartz veinlets passing through the chalcopyrite and tetrahedrite are usually clean cut, and form connecting lines between the quartz in the pyrite rosettes. In places, however, the quartz veins enlarge and project into the chalcopyrite and tetrahedrite in such an irregular way as to suggest replacement:

Some of the areas in the tetrahedrite might be regarded as contemporaneous with the sulphide were it not for their connection with the feeding channels.

It seems to the writer that the structural conditions and mineral relationships described above lead necessarily to the following conclusions:

Following the deposition of the sulphides, fracturing of the ore occurred, resulting in the introduction of silica that was deposited in the chalcopyrite and tetrahedrite largely as fracture filling, but in the chalcopyrite at least in part by replacement. In addition it was deposited in the pyrite almost wholly by replacement.

The reasons for believing the above to be the case are that:

1. In every specimen examined the pyrite seemed to be earlier than the chalcopyrite, or in some cases contemporaneous with it.

2. The quartz forms veinlets which cut all the sulphides, thus precluding the possibility of replacement of quartz by pyrite or the other sulphides.

3. Clearly defined channels or feeders of quartz often lead from one pyrite rosette to another, showing the lines along which the silica-bearing solutions entered the ore.

4. The pyrite rosettes show all stages in the replacement process, but curiously enough the change seems to have worked from the center outward.

5. It does not seem likely that the pyrite could have been introduced with the quartz, otherwise one would expect to find deposition of pyrite along the feeding channels, whereas this is not the case.

6. The development of the crystal outlines of quartz in the chalcopyrite in places, indicates clearly a replacement of the chalcopyrite by the quartz, and the association of these with the feeding channels shows the later introduction of the material.



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## The Southern Extremity of the "Clinton" Gas Pools in Ohio

BY L. S. PANTITY,\* COLUMBUS, O.

(St. Louis Meeting, October, 1917)

THE Cleveland (O.) gas pool described by Frank R. Van Horn,<sup>1</sup> is the northern extremity of the great "Clinton" sand gas development in

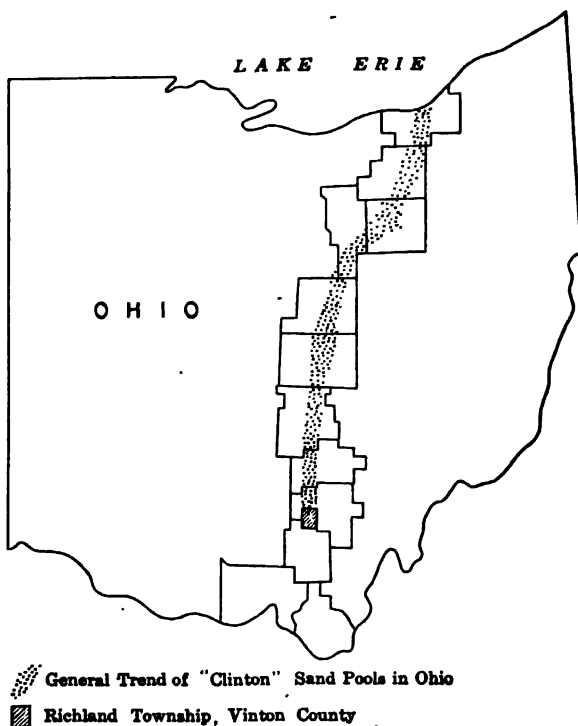


FIG. 1.

Ohio. Numerous gas pools have been developed in this formation, which are now more or less connected, link by link, forming an almost continuous gas pool, starting at Cleveland, and running thence southward through Cuyahoga, Medina, Wayne, Ashland, Richland, Knox, Licking,

\* Geologist, Ohio Fuel Supply Co.

<sup>1</sup> *Bulletin* No. 121 (January, 1917).

Fairfield, Hocking and Vinton counties. The present southern extremity of these connected pools is in Richland Township, Vinton County (Fig. 1).

The writer offers his analysis of the Richland Township development, which may serve to compare the northern and southern extremities.

### *Geology*

The geologist is greatly handicapped by the lack of a suitable key-horizon, inasmuch as the outcropping rocks are members of the "Big Injun" series of the Logan Group (Mississippian). The underlying shales (Waverly shales) show that considerable erosion took place before the deposition of the "Big Injun," which accounts for the presence of an erosion plane between these two formations, further complicating the study of the geological structure from surface outcrops.

A reliable structural map of the "Clinton" is obtainable only by spirit-level elevations at the mouth of the various wells drilled, and the elevation of the sand calculated from the logs of the wells. The accompanying isobath map has been prepared in this manner (Fig. 2). Since the completion of this map, numerous wells have been drilled in the township, which, if taken into consideration might cause a slight variation in the appearance of the contour lines. The monotonous monoclinal structure of the "Clinton" sand is evident from this map.

A vertical section of the formations penetrated by the drill may be constructed from the following table:

|  |                        |
|--|------------------------|
| "Big Injun" and Waverly shales.....              | In surface outcrops.   |
| Berea Sandstone.....                             | 20 to 80 ft. thick     |
| Bedford (Carboniferous) and Ohio (Devonian)..... | 750 to 900 ft. thick   |
| "Big Lime" (Corniferous and Niagara).....        | 570 to 675 ft. thick   |
| Interval "Big Lime" to "Clinton".....            | 175 to 210 ft.         |
| Interval "Clinton" to Trenton .....              | 1,350 ft. (estimated). |

The thickness of the Berea is variable from place to place without any regularity; but in the other formations cited, the smaller measurements are met with in the western part of the township, the thickening taking place toward the east (Fig. 3).

The Berea lies at an elevation of about 250 ft. above sea level at the western line, and is close to sea level at the eastern line of the township. The average dip for the Berea is about 35 ft. per mile. This compares with an average dip of 80 ft. per mile for the "Clinton" in the same territory. The difference in dip is due to the lateral variation of the intervening strata.

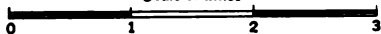
Although the "Clinton" sand development has shown us but few pronounced anticlinals, there remains a great deal of unstudied territory where the pools may be associated with some sort of folding. Work is

now under way in another section of the State, where anticlinal structure is expected.



STRUCTURAL MAP OF THE "CLINTON" SAND

Scale of Miles



CONTOUR INTERVAL 100 FEET  
STRUCTURE CONTOUR 50 FEET

DATUM - MEAN SEA LEVEL  
DATUM - 2000 BELOW SEA LEVEL

FIG. 2.

### *Oil and Gas*

Oil and gas have been struck in various geologic horizons; the first one of which, in descending order, is the Berea. A gas pool is in operation from this formation in the southeast quarter of Richland Township, and

some scattered wells producing gas from the Berea are found south of Richland Township. The Berea contains water in this territory.

Some gas has been encountered in the Ohio shales, and one or two wells produce a little oil from the top of the "Big Lime" (which is probably the Ragland sand of Kentucky), but so far no commercial pool has been developed below the Berea until the "Clinton" is reached.

South of Richland Township but two wells are producing gas from the "Clinton" and they are in the immediate vicinity of the southern township line. Numerous tests have been drilled in a southerly direction as far as the Ohio River, but so far, without encouraging results to the operators.

The gas pools in the township are described by the driller as very "spotted." All observed conditions indicate lenticular sand bodies; and differential cementation has been a great factor in the accumulation

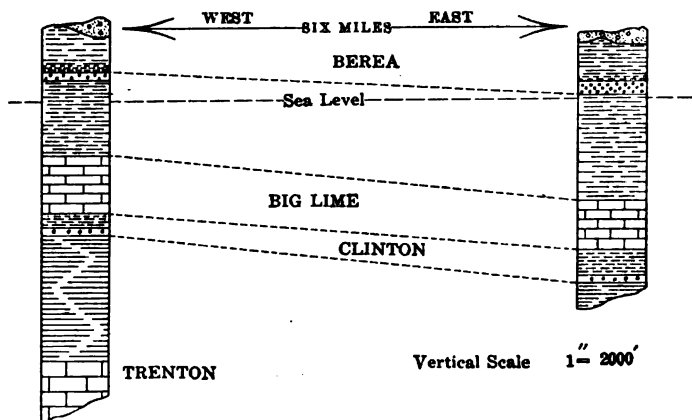


FIG. 3.

of gas along this monoclinel. Another important element under consideration is the fact that connate water is rarely met. The stratum is considered by the driller, free from water. No doubt in further tests in localities where the "Clinton" lies at a greater depth below sea level, connate water will be encountered, when, of course, structural conditions will be of great importance.

In following up "wildcat" wells, tests located along the strike of the formation have been more successful than those located otherwise.

The initial open flow volume of wells is about two to four million cubic feet per day. The largest open flow measurements showed eight to nine million cubic feet of gas per day, when they were drilled in. The rock (or closed) pressure of the wells is between 650 and 750 lb. per square inch.

It is rather early to make any statements regarding the life of these

wells, but no doubt the wells will produce for a greater length of time than did those in Cleveland, inasmuch as the large acreage at the disposal of the operators will enable a more judicious spacing of the wells, and will prevent the recurrence of the overcrowded condition which characterized the Cleveland town-lot excitement.

No producing horizon below the "Clinton" has yet been found. The Trenton limestone, which is the producing horizon of northwestern Ohio, is estimated to be about 1,350 ft. below the "Clinton." The depth at which this stratum would be reached in this territory would be about 3,500 to 4,000 ft. The nearest wells to this locality that have reached the Trenton are at or near the towns of Waverly, Ironton and Oak Hill. In all cases the Trenton was found barren of oil or gas. The chemical analyses of the Trenton in these wells are not known to the writer, but it is doubtful whether the magnesium contents were large enough to create a sufficiently porous rock which might serve as a reservoir for petroleum. In a well drilled through the Trenton into the St. Peter sandstone, north of Mount Vernon (O.) the Trenton was found at a depth of 3,645 to 4,515 ft. A chemical analysis showed the magnesium carbonate contents of the Trenton in that well as follows:

| Depth, Feet          | MgCO <sub>3</sub><br>Per Cent. |
|----------------------|--------------------------------|
| 3,645 .....          | 3.94                           |
| 3,718 to 3,723 ..... | 3.91                           |
| 3,723 to 3,728 ..... | 5.19                           |
| 3,728 to 3,733 ..... | 4.16                           |
| 3,733 to 3,742 ..... | 3.81                           |

When comparing these figures with the figures obtained from the analyses of the Trenton at Lima (O.), where it is a prolific source of oil and gas, and the magnesium carbonate contents range from 25 to 45 per cent., the prospects for a Trenton pool in this territory appear to be very slight.

The present eastern extension of the Trenton oil production is about 2 mi. west of Caledonia, Marion County, where a small producer has been recently drilled and oil found 585 ft. in the Trenton, the few feet of pay sand giving the following chemical analysis: SiO<sub>2</sub>, 17.02 per cent.; CaCO<sub>3</sub>, 46.65 per cent.; MgCO<sub>3</sub>, 25.75 per cent.; Balance, 10.58 per cent.

This well has created the usual excitement and further work is now contemplated in an attempt to open up a Trenton pool in that locality.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper

## Graphic Solutions of Some Compressed-Air Calculations

BY C. W. CRISPELL,\* NEW HAVEN, CONN.

(St. Louis Meeting, October, 1917)

THE four nomograms presented in this article were designed to simplify and make more rapid the calculations connected with the compression and transmission of air. The formulæ involved are rather complicated, but the graphic solutions are simple; their use requires an ability to follow explicit directions and to interpolate values on a logarithmic scale.†

For the sake of convenience, the nomograms will be divided into two classes, and the purpose and accuracy of each will be discussed.

### *Compression*

It is frequently desirable to determine the horsepower necessary to compress a given amount of free air from atmospheric pressure to a certain final pressure. The nomograms in Figs. 1 and 2 present a rapid method of calculating the horsepower required to compress 1 cu. ft. of free air per minute by single or two-stage compression. The general formula for the horsepower required to compress air adiabatically is:

$$HP = \frac{S \times 144}{33,000} \cdot \frac{PVn}{(n-1)} \left[ \left( \frac{P'}{P} \right)^{\frac{n-1}{n}} - 1 \right] \dots \dots (1)$$

where:

$P$  = absolute atmospheric pressure at compressor, in pounds per square inch.

$P'$  = final absolute pressure of compressed air, in pounds per square inch.

$V$  = volume of free air compressed per minute, in cubic feet.

$S$  = number of stages.

$n$  = exponent of the compression curve.

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\* Fifth-year student in Mining Engineering, Hammond Laboratory, Sheffield Scientific School, Yale University.

† In order to meet publication requirements, the nomograms have been reproduced on a reduced scale and are intended merely to illustrate the method of determining compressed-air data, but separate large-scale plates have been prepared, suitable for working charts, which members may obtain free of charge by addressing the Secretary.





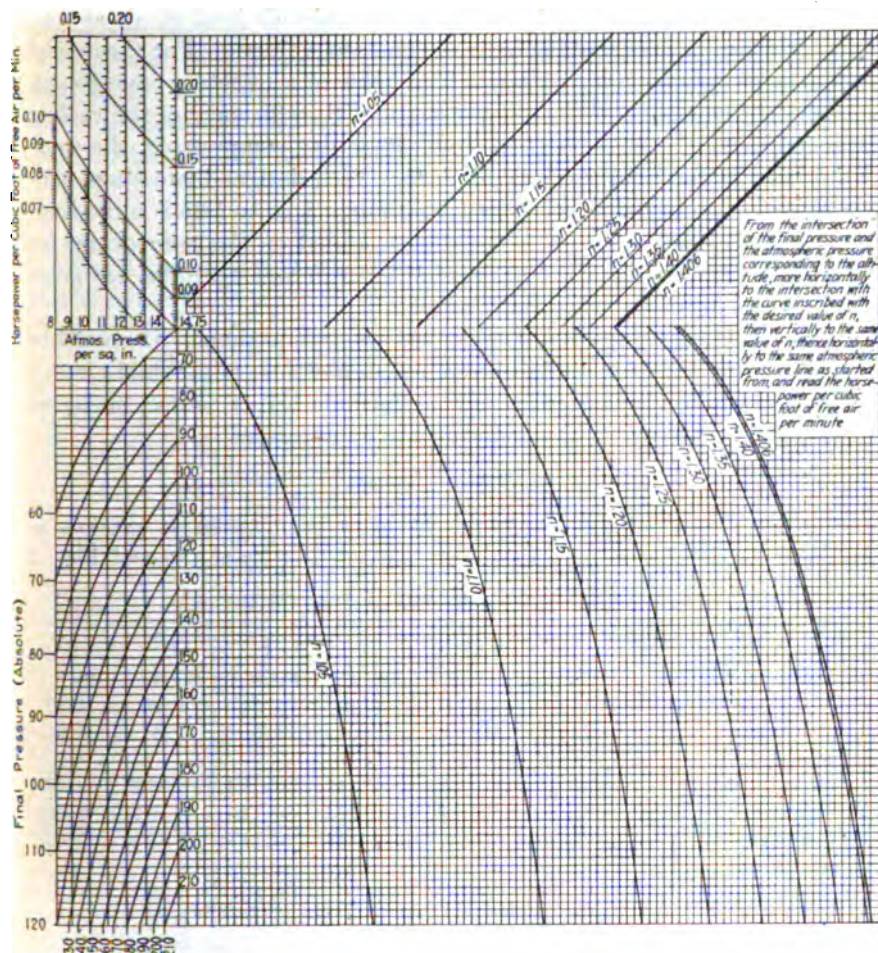


FIG. 2.

| Altitude in Feet | Barometric Pressure |                    | Altitude in Feet | Barometric Pressure |                    |
|------------------|---------------------|--------------------|------------------|---------------------|--------------------|
|                  | Inches, Mercury     | Pounds per Sq. In. |                  | Inches, Mercury     | Pounds per Sq. In. |
| 0                | 30.00               | 14.75              | 8,000            | 22.11               | 10.87              |
| 1,000            | 28.80               | 14.20              | 9,000            | 21.29               | 10.46              |
| 2,000            | 27.80               | 13.67              | 10,000           | 20.49               | 10.07              |
| 3,000            | 26.76               | 13.16              | 11,000           | 19.72               | 9.70               |
| 4,000            | 25.76               | 12.67              | 12,000           | 18.98               | 9.34               |
| 5,000            | 24.79               | 12.20              | 13,000           | 18.27               | 8.98               |
| 6,000            | 23.86               | 11.73              | 14,000           | 17.59               | 8.65               |
| 7,000            | 22.97               | 11.30              | 15,000           | 16.93               | 8.32               |

The exponent  $n$  is the ratio between the specific heat of air at constant pressure and its specific heat at constant volume. For theoretical adiabatic compression  $n = 1.406$ . The value of  $n$  in actual compression varies with the design of the compressor and the efficiency of the cooling arrangements. In well-designed single-stage dry compressors, the value of  $n$  ordinarily lies between 1.25 and 1.35. Well-designed two-stage compressors give values of  $n$  as low as 1.15, though 1.20 to 1.25 is more common.<sup>1</sup> Additional values of  $n$  have been added on the nomograms for the sake of completeness. Values of  $P$  for different altitudes are given in the tables accompanying Figs. 1 and 2.

Directions for using Figs. 1 and 2 are printed on them. It should be noted that the scale for the horsepower per cubic foot of air starts from a different ordinate for each value of  $P$ . Formula (1), of course, makes no allowance for the mechanical losses in the compressor.

### *Speed and Accuracy*

To test the diagrams, six problems have been taken at random as shown in Tables 1 and 2. They were solved first by the use of Figs. 1

TABLE 1.—*Single-Stage Compression*

| Problem No.....      | 1     | 2     | 3     | 4     | 5     | 6     |
|----------------------|-------|-------|-------|-------|-------|-------|
| Final pressure.....  | 70.0  | 60.0  | 90.0  | 80.0  | 50.0  | 40.0  |
| $n$ .....            | 1.30  | 1.35  | 1.40  | 1.25  | 1.15  | 1.20  |
| Atmos. pressure..... | 14.7  | 13.0  | 12.0  | 11.0  | 10.0  | 8.0   |
| Horsepower:          |       |       |       |       |       |       |
| By chart.....        | 0.122 | 0.108 | 0.144 | 0.118 | 0.079 | 0.066 |
| By slide rule.....   | 0.121 | 0.107 | 0.143 | 0.116 | 0.078 | 0.065 |
| By logs.....         | 0.120 | 0.107 | 0.143 | 0.117 | 0.078 | 0.065 |
| Per cent. error:     |       |       |       |       |       |       |
| By chart.....        | +1.60 | +0.93 | +0.70 | +0.85 | +1.28 | +1.54 |
| By slide rule.....   | +0.80 | 0.0   | 0.0   | -0.85 | 0.0   | 0.0   |

TABLE 2.—*Two-Stage Compression*

| Problem No.....          | 1     | 2     | 3     | 4     | 5     | 6     |
|--------------------------|-------|-------|-------|-------|-------|-------|
| Final pressure.....      | 150.0 | 130.0 | 110.0 | 100.0 | 80.0  | 60.0  |
| $n$ .....                | 1.40  | 1.35  | 1.30  | 1.25  | 1.20  | 1.15  |
| Atmos. pressure.....     | 14.7  | 14.0  | 13.0  | 12.0  | 10.0  | 9.0   |
| Horsepower:              |       |       |       |       |       |       |
| By chart.....            | 0.177 | 0.159 | 0.139 | 0.125 | 0.100 | 0.080 |
| By slide rule.....       | 0.175 | 0.158 | 0.138 | 0.125 | 0.099 | 0.079 |
| By logs.....             | 0.175 | 0.158 | 0.138 | 0.124 | 0.099 | 0.079 |
| Per cent. error:         |       |       |       |       |       |       |
| By chart.....            | +1.14 | +0.63 | +0.72 | +0.80 | +1.01 | +1.26 |
| By slide rule.....       | 0.0   | 0.0   | 0.0   | +0.80 | 0.0   | 0.0   |
| Mean error by chart..... | 0.93% |       |       |       |       |       |
| Mean error by rule.....  | 0.13% |       |       |       |       |       |

<sup>1</sup> R. Peele: *Compressed-Air Plant*, Wiley, N. Y., 1913.

and 2 and then by means of a 20-in. slide rule applied to formula (1). The average time required to solve a problem with the diagrams was 1 min. 45 sec. as compared to 3 min. 45 sec. for the slide rule. The mean error in these calculations, disregarding + and - values, was 1.15 per cent. for the graphic solution and 0.28 per cent. for the slide rule.

### Transmission

Fig. 3 gives a graphic solution of D'Arcy's formula for the transmission of compressed air in pipes. This formula is:<sup>2</sup>

$$D = c \sqrt{\frac{d^5 (p_1 - p_2)}{w_1 l}} = \frac{c \sqrt{d^5}}{\sqrt{l}} \sqrt{\frac{(p_1 - p_2)}{w_1}} \dots \dots (2)$$

where:

- $D$  = the volume of compressed air in cubic feet per minute discharged at the final pressure.
- $c$  = a coefficient varying with the diameter of the pipe, as determined by experiment.
- $d$  = actual diameter of pipe in inches.
- $l$  = length of pipe in feet.
- $p_1$  = initial gage pressure in pounds per square inch.
- $p_2$  = final gage pressure in pounds per square inch.
- $w_1$  = the density of the air, or its weight per cubic foot, at initial pressure.

The nomogram, Fig. 3, allows the solution of this equation without reference to tables for values of  $c \sqrt{d^5}$  and  $\sqrt{\frac{p_1 - p_2}{w_1}}$  which are ordinarily used for this purpose.

The following cases may arise in connection with the transmission of compressed air through pipes:

1. Given the volume of compressed air, the length of pipe, the initial pressure, and the maximum allowable drop in pressure; required, the diameter of the pipe.

2. Given the length of pipe, the diameter of pipe, the initial pressure, and the maximum allowable drop in pressure; required, the volume of compressed air which the pipe will carry.

3. Given the diameter of the pipe, the volume of compressed air, the initial pressure, and the maximum allowable drop in pressure; required, the maximum length of pipe of the given diameter which can be used under these conditions.

4. Given the diameter of the pipe, the volume of compressed air, the length of pipe, and the maximum allowable drop in pressure; required, the pressure at which the air must enter the pipe.

<sup>2</sup> R. Peele: *Op. cit.*

5. Given the diameter of the pipe, the volume of compressed air, the length of pipe, and the initial pressure; required, the drop in pressure.

The graphic solution of each of these problems may be obtained from Fig. 3 as follows:

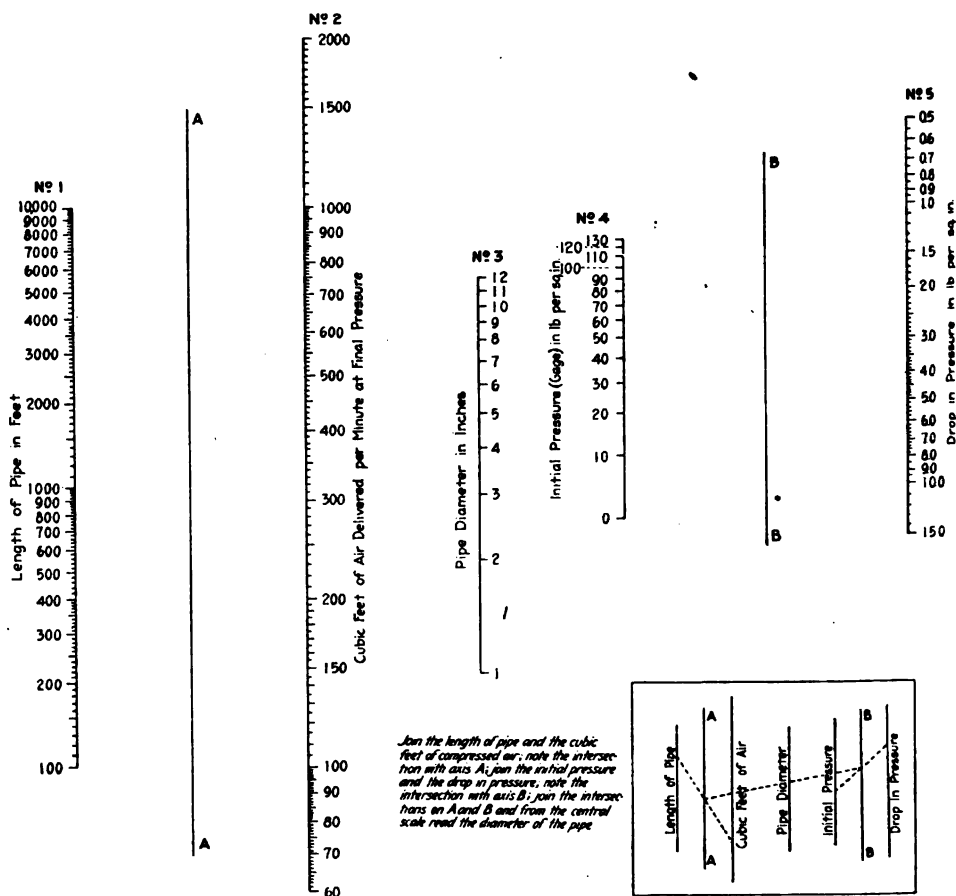


FIG. 3.

**Case 1.**—With a straight edge, join the length of pipe and the cubic feet of compressed air (not free air); note the intersection on axis A; join the initial pressure with the drop in pressure; note the intersection on axis B. A line joining the two points on A and B will intersect scale number 3 at the required pipe diameter.

**Case 2.**—With a straight edge, join the initial pressure with the drop in pressure, note the intersection on axis B; join the intersection on B with the diameter of the pipe; note the intersection on axis A. A line joining this point on A with the length of pipe will intersect scale number 2 at the required volume of compressed air.

**Case 3.**—With a straight edge, join the initial pressure with the drop in pressure; note the intersection on axis *B*; join the intersection on *B* with the diameter of the pipe; note the intersection on axis *A*. A line joining this point on *A* with the given cubic feet of compressed air intersects scale number 1 at the required maximum length of pipe.

**Case 4.**—With a straight edge, join the length of pipe with the volume of compressed air; note the intersection on axis *A*; join the intersection on *A* with the diameter of the pipe; note the intersection on axis *B*. A line joining this point on *B* with the allowable drop in pressure intersects scale number 4 at the required initial pressure at the entrance of the pipe.

**Case 5.**—With a straight edge, join the length of pipe with the volume of compressed air; note the intersection on axis *A*; join the intersection on *A* with the diameter of the pipe; note the intersection on axis *B*. A line joining this point on *B* with the given initial pressure will intersect scale number 5 at the required drop in pressure.

### *Speed and Accuracy*

The six problems given in Table 3 were worked out to test this nomogram. An average time of 1 min. 45 sec. was required to solve a problem using a 20-in. slide rule in connection with the tables published in *Peele's Compressed-Air Plant*. These same problems were solved by the nomogram in an average time of 1 min. The mean error of the nomogram is less than 0.5 per cent. as shown by Table 3.

TABLE 3.—*D'Arcy's Formula*

| Problem No.....       | 1     | 2     | 3     | 4     | 5     | 6       |
|-----------------------|-------|-------|-------|-------|-------|---------|
| Size of pipe.....     | 2     | 4     | 6     | 8     | 10    | 12      |
| Length of pipe.....   | 100   | 1,000 | 2,000 | 3,000 | 4,000 | 5,000   |
| Initial pressure..... | 40    | 60    | 70    | 80    | 90    | 100     |
| Drop in pressure..... | 2.0   | 2.0   | 3.0   | 4.0   | 5.0   | 8.0     |
| Cubic feet of air:    |       |       |       |       |       |         |
| By chart.....         | 78.2  | 133.0 | 309.0 | 574.0 | 930.0 | 1,600.0 |
| By slide rule.....    | 78.8  | 133.6 | 308.2 | 573.0 | 935.0 | 1,603.0 |
| By logs.....          | 78.9  | 133.4 | 308.3 | 573.0 | 935.0 | 1,603.0 |
| Per cent. error:      |       |       |       |       |       |         |
| By chart.....         | -0.88 | -0.30 | +0.22 | +0.17 | -0.53 | -0.18   |
| By slide rule.....    | -0.12 | +0.15 | -0.03 | 0.0   | 0.0   | 0.0     |

Many of the data on the capacity of air compressors and the air consumption of rock drills or other machines are expressed in terms of cubic feet of free air. The volume of compressed air to be transmitted must be known before the nomogram for D'Arcy's formula, Fig. 3, can be used. Volumes of free air at different altitudes can be converted into corre-

sponding volumes of compressed air at different absolute pressures by means of nomogram on Fig. 4. This diagram is based on the equation:

$$pv = p'v'$$

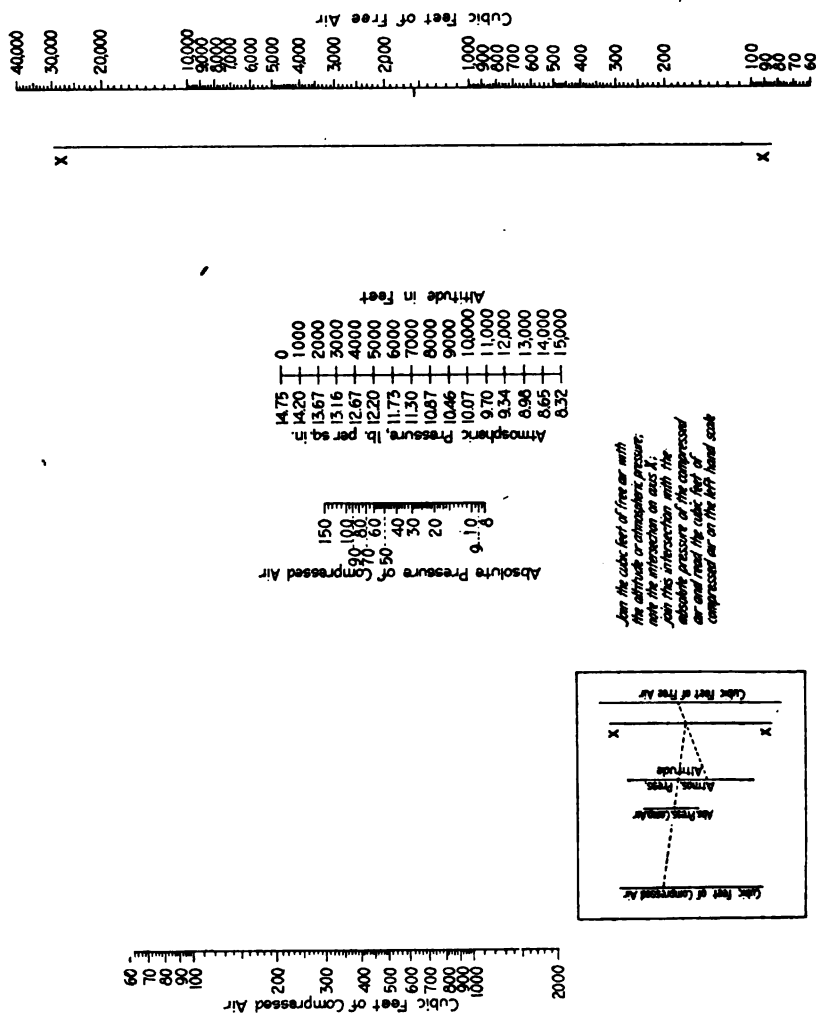


FIG. 4.

where:

$p$  = initial absolute (atmospheric) pressure in pounds per square inch.

$v$  = initial volume of air in cubic feet.

$p'$  = final absolute pressure of compressed air in pounds per square inch.

$v'$  = volume of compressed air in cubic feet.

The accuracy of this chart as compared to calculations with a 20-in. slide rule and with logarithms is shown by Table 4.

TABLE 4

| Problem No. ....            | 1     | 2     | 3     | 4     | 5     | 6      |
|-----------------------------|-------|-------|-------|-------|-------|--------|
| Altitude.....               | 0     | 1,000 | 2,000 | 3,000 | 6,000 | 14,000 |
| Press. of comp. air.....    | 20    | 40    | 60    | 80    | 90    | 100    |
| Cubic feet of free air..... | 100   | 500   | 1,000 | 1,500 | 2,000 | 3,000  |
| Cubic feet of comp. air:    |       |       |       |       |       |        |
| By chart.....               | 74.0  | 178.0 | 227.0 | 248.0 | 260.0 | 259.0  |
| By slide rule.....          | 73.8  | 177.5 | 228.0 | 247.0 | 261.0 | 259.5  |
| By logs.....                | 73.8  | 177.5 | 227.8 | 246.8 | 260.7 | 259.5  |
| Per cent. error:            |       |       |       |       |       |        |
| By chart.....               | +0.27 | +0.28 | -0.35 | +0.49 | -0.26 | -0.15  |
| By slide rule.....          | 0.0   | 0.0   | +0.09 | +0.08 | +0.11 | 0.0    |

The figures given above for speed of work with Figs. 1, 2, and 3, as compared with the slide-rule work, do not indicate the true value of the nomograms. The problems were worked out at a time when the author had been using equations (1) and (2) and was entirely familiar with their terms and their algebraic solution. At a later date, when he had become somewhat rusty perhaps both in his mathematics and his knowledge of compressed air, the graphic solution would take relatively much less time and be more apt to give numerically correct results.

In conclusion, the author desires to give credit to Prof. H. L. Seward for many helpful suggestions regarding the construction of the nomograms and to Prof. J. F. McClelland for kind assistance in the preparation of this article.





DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Practical Value of Oil and Gas Bureaus

BY W. G. MATTESON,\* B. S., E. M., E. MET., HOUSTON, TEX.

(St. Louis Meeting, October, 1917)

THE Oklahoma legislature recently passed a bill providing for "the creation of an oil and gas department under the jurisdiction of the Corporation Commission, authorizing the Corporation Commission to appoint a chief oil and gas conservation agent, and conferring exclusive jurisdiction on the Corporation Commission in reference to the conservation of oil and gas, and the inspection of gasoline and oil, the product of crude petroleum, and repealing all acts or parts of acts in conflict therewith and declaring an emergency." Some of the functions of the new bureau include the receiving and filing of all well logs and the direction and supervision of plugging all abandoned oil and gas wells under rules prescribed by the Corporation Commission.

The two leading petroleum-producing States of the country, Oklahoma and California, have now established oil and gas bureaus with a staff of inspectors to see that the important rules relating to the conservation of oil and gas are thoroughly enforced. The economic significance and the value of well logs in the conservation work has been recognized and the necessary legal action taken to insure and procure them. William B. Heroy, of the Government Land Classification Board, in a personal communication to the writer, says that the drilling of wells is also regulated by public authority in the Roswell artesian basin in southeastern New Mexico, where a permit must be obtained for the drilling of any well and a certified log of the well recorded with a county official. The method is said to give fairly satisfactory results.

In a recent article<sup>1</sup> a National instead of a State Bureau was suggested for collecting and filing well logs. In discussing the paper, L. L. Hutchison characterized such a plan as impractical because the average driller or contractor will often juggle or guess at the log and will not keep the record sufficiently close or accurate, making it necessary to maintain a special man at each well for the sole purpose of obtaining

\* Petroleum geologist and mining engineer, Producers Oil Co.

<sup>1</sup> W. G. Matteson: The Need and Advantages of a National Bureau of Well Log Statistics, *Bulletin* No. 122 (February, 1917).

such data correctly, a procedure that is expensive. A number of geologists and producers will take issue with this contention. Moreover, the excellent logs of every well in California, obtained at little or no expense, form the best refutation and answer to such an objection. It should be further borne in mind, as suggested previously, that a log need not be accurate in a strict scientific sense to be of great practical value when projected on the scale of 1 in. = 100 ft. Where a number of logs are obtained from the same field, any glaring inaccuracies resulting from an improper record are generally detected when checked against the average of several adjacent logs. Such inaccurate logs can therefore be rejected.

To emphasize further the practical value of well logs from a geological standpoint, the writer knows of a company that has saved and made several millions of dollars in one pool by the judicious application of such data. This company industriously gathered well records for a considerable period and projected them on a scale of 1 in. = 100 ft. In this way, the structure of the pool was completely and quite accurately defined, the purchase of much high-priced but poor acreage avoided, while acreage that was considered questionable was shown to contain some of the best wells in the region. All the large producing oil companies have recognized the practical importance of well logs and many now maintain special subsurface departments whose chief functions are the recording and plotting of such data.

As for the difficulty of getting the head driller or contractor to keep an accurate record of the well, the writer has questioned a number of operators who say that such difficulty is greatly exaggerated. The driller generally shows a tendency to cooperate in every way and to obtain that which is requested. In some regions, it might be necessary to educate the driller to the point of recognizing the value of accurate logs. In this connection, it is gratifying to note the vast improvement during the last 2 to 4 years in this matter. Where there were 10 accurate records 5 years ago, there are a thousand today. The "oil smeller" or geologist is no longer viewed with suspicion or contempt by the field men; he has come into his own. If such results can be secured under very unfavorable conditions, there seems to be no logical reason why a more determined and organized effort, supported by public authority and directed by a standard and efficient bureau, should not produce the results desired. Aside from the geological importance, if water troubles are to be prevented, if desirable and satisfactory data are to be obtained on the life and rate of decline of wells, if accurate estimates of oil reserves are to be obtained, well logs are vital and an efficient oil and gas bureau becomes a practical necessity.

Despite the advantages that a National bureau of well-log statistics or a National oil and gas bureau might possess over a State organization of similar design, the establishment of such a bureau would constitute

undoubtedly an infringement of State rights.<sup>2</sup> Herein lies the main obstacle to the plan for a National organization and there seems to be no way at present to overcome it. The matter, therefore, must await the initiative of each State, and it is to be hoped that the importance of such an organization will be so recognized that the bureau will not be made ineffective by the appointment, for political reasons, of incompetent and inexperienced officials.

The new Oklahoma Oil and Gas Bureau has unlimited possibilities, since its powers are not expressly defined by any definite and prescribed statute as they are in California. The Oklahoma bureau is under the direct jurisdiction and supervision of the Corporation Commission, a body which has the power to hear and to sit in judgment on any case or complaint pertaining to oil and gas and their conservation, and to prescribe and cause to be enforced any rule or provision which it deems necessary to remedy an undesirable condition. The development of this bureau will therefore be studied with interest. Meanwhile it would be gratifying to see similar State bureaus established in Texas, Louisiana, Pennsylvania, West Virginia, Ohio, Illinois, Wyoming, Kentucky and all other States where extensive drilling operations are now in progress.

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<sup>2</sup> M. L. Requa: Discussion of the Need and Advantages of a National Bureau of Well-Log Statistics, *Bulletin* No. 124 (April, 1917), 635.



## Problems Connected with the Recovery of Petroleum from Unconsolidated Sands

Discussion of the paper of WILLIAM H. KOBÉ, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 120, December, 1916, pp. 2253 to 2276.

WILLIAM H. KOBÉ (communication to the Secretary\*).—I have read with much interest the discussion of my paper by Arthur Knapp and I. N. Knapp published in the March *Bulletin*.

These discussions apparently assume that my paper advances a number of ideas that are not intended:

1. That the total sandbody or pay is to be removed.
2. That upon bringing this sand and oil to the surface 100 per cent., or perfect separation, can take place.
3. That this tremendous quantity of sand is then hauled away in wagons and distributed as road material.

The paper under discussion is not a theoretical treatment of the subject but is the result of actual experience in oil production under the conditions mentioned. That wells that produce sand produce more oil, and the best operating methods to bring this about are fully treated. It is, of course, impossible to remove all the oil sand underlying a property, but the greater the amount removed the greater becomes the production of oil.

As stated in my paper, the surface disposal of this large volume of sand is a problem and the only use to which it can be put is road surfacing, but I did not intend to convey the idea that all this sand was thus used. Only a very small quantity was required on the company's roads, and two wagons only were employed for this purpose. The remaining sand accumulated in the numerous "washes" and sumps.

Although this sand contained sufficient oil to stain the hands, it was surprisingly dry and could be easily shoveled. As stated in the paper, "the piles of sand are gradually freed of their oil content and a crust forms on their surface which will bear the weight of a man." I consider this a high degree of separation. A small amount of oil necessarily remains in the sand and is held there by capillarity. Otherwise it would be useless as a road binder, and, furthermore, to recover such oil special processes of treating are required.

Arthur Knapp says, "of the total volume produced by pumping, not more than 4 or 5 per cent. can possibly be sand." I cannot agree with this statement as I have managed many properties where the wells were so operated that the amount of sand produced was five times this figure.

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\* Received, Apr. 29, 1917.

The pumps used for this purpose were Axelson extension barrels with steel plungers and steel-packed standing valves. Sand cannot be handled with the usual Eastern pumping equipment and experience under local conditions is necessary to operate sandy wells, maintain production and prevent the pumps from "sanding up."

I. N. Knapp takes exception to the statement that "other things being equal, the maximum recovery of oil from an unconsolidated sand is directly dependent upon the maximum recovery of the sand itself" and asks what other things must be equal. The meaning of this statement is simply that the more sand a well produces, the greater is the ultimate amount of oil recovered. In order to compare two or more wells, the "other things that must be equal" are naturally gas pressure, viscosity of the oil, depth and physical aspect of the sand, and the age of the wells—in other words, it would be useless to arrive at conclusions by comparing the production of wells operating under radically different physical conditions. It is maintained that if two wells of about the same age are producing 15 Beaumé oil from the same horizon of unconsolidated sand under like conditions of gas pressure, depth, etc., the one which produces sand has the greater output of oil.

I agree with I. N. Knapp that from a theoretical standpoint the prevention of sand entry and caving appears to be the correct procedure, but my experience is that wherever this has been followed the wells suffered a marked decline in production. I believe in the "regulated entry" of sand and the use of a screen mesh suited to this purpose. Fine sand has not troubled my pumps as much as small gravel and stones.

I. N. Knapp further says that "if a proper mesh of screen has been employed, the extraction of the oil when properly controlled cannot induce caving. The proper screening of a well reduces pump troubles to a minimum, always assuming that the well may become a pumper." I agree fully with this statement, but hold that under such conditions the well will not produce the maximum amount of oil possible.

The occasional injury to casing from shifting sand is not fatal to the well. I have redrilled many such wells, and it is not unusual for a well to continue producing great quantities of sand and oil after the initial flexure of the casing occurs. It requires a serious injury to so deflect the casing that it becomes impossible to tube it.

I realize that redrilling is expensive; that sand production is very troublesome; that underground caving frequently follows sand removal and that all of these conditions may be prevented, but I maintain that such prevention is invariably at the expense of oil production. I know of a great many wells that have produced large volumes of sand but which continue to pump with no indications of casing damage.

A. B. Thompson, in his recent book on oil-field development, describes the use of screen casing at some length and concludes that although

such casing may be utilized to prevent sand entry, in practice this results in decreased oil production. He advocates sand removal.

## The Influence of the Movement of Shales on the Area of Oil Production

Discussion of the paper of RICHARD A. CONKLING, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 1960 to 1972.

RICHARD A. CONKLING (communication to the Secretary\*).—Mr. Hager says that his results on the correlation of well logs in the north Cushing field are the opposite to those of the author. Can he, then, explain why the structural dip of the formations increases with depth, as Dr. Ohern, in his criticism, sets forth that it does, and as all who have worked in these fields will admit, if the shales thicken on the crest of the anticline, as he says they do? We are willing to agree with Dr. Ohern that there is some evidence of a slight unconformity below the Bartlesville sand and think that we can prove, from a specimen in our office, that this unconformity comes at the base of the Bartlesville sand in the south Cushing pool.

On the Ingalls dome, in the southeast corner of Township 19 North, Range 4 East, however, we get results the same as those given by the author on Cushing. The Ballard well in the center of the Northeast Quarter of Section 33 is just 10 ft. higher, structurally, according to the surface contour, than the Ringold well in the center of the Southwest Quarter of Section 34. The dip in the Bartlesville sand, between these wells, is 40 ft. and the dip on the top of the Mississippi lime is 75 ft. This also might be accounted for, partly by the unconformity, but the author used the interval between the top of the Oswego lime, which is about 420 ft. above the Bartlesville sand, and the oil horizon found in the lime in the Ballard well—just mentioned above—to find the depth at which the same oil should be found in a well in Section 5, Township 22 North, Range 3 East, 25 miles North. Using this interval below the Oswego in the former well, we predicted within 2 ft. where they would find the oil in the latter well. Of course, we know that it was accidental that we missed it only by 2 ft., but if there were much of an unconformity in this interval it would be more of an accident if we came within 100 ft. of the depth. Since it can be proven from most of the structures in this section that the dip continues to increase downward, below the Bartlesville at least, to the Mississippi lime, it does not seem necessary to the author to try to prove otherwise that the depth at which the shales will tend to thicken on top of a fold, has not been reached above the

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\* Received, Apr. 20, 1917.

Tucker sand, if, as Dr. Ohern says, there is a certain place where such will be the case.

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### The Need and Advantages of a National Bureau of Well-Log Statistics

Discussion of the paper of W. G. MATTESON, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 287 to 290.

W. G. MATTESON (communication to the Secretary\*).—The criticism and question raised by Arthur Knapp regarding the standardization of nomenclature is most important. Faulty rock classification is generally the result of carelessness or ignorance on the part of the driller. Two remedies might be suggested. In a field where several large companies are operating on an extensive scale, a geologist from each concern might be assigned to superintend the gathering of well-log data and the classification of samples. The geologists from the several operating companies might then combine their data, which would doubtless give valuable information to all interested parties. At present, coördination is not a big factor, unfortunately, but the time will come when conservation will demand such systematic coördination on the part of competing companies. The second remedy lies in instructing and educating the driller to a thorough understanding of the importance of an accurate log and correct classification of the various formations.

The writer does not agree with the statement that "90 per cent. of the logs as taken today are absolutely worthless to any one but the one who has taken the log." While mistakes in nomenclature often occur, the driller is generally able to classify the important water-, oil-, and gas-bearing formations with a fair degree of accuracy. Such information is often sufficient to permit accurate correlation and also constitutes valuable data for projection purposes. It must be borne in mind that to attempt to use well-log data in a strict scientific sense of accuracy is impracticable and impossible. A broad interpretation and a comprehensive knowledge of the practical limits of such information are essential in applying it successfully.

Even experienced geologists do not always agree on rock classification. The question as to what constitutes correct nomenclature should receive the earnest and immediate attention of geologists, operators, and producers. Although standard textbooks on geology give correct definitions and descriptions of the general rock species, such textbooks fail to develop the subject in sufficient detail for the work under consideration. A committee composed of appointed members from the American Institute of Mining Engineers, the Geological Society of America, and the

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\* Received Apr. 23, 1917.



U. S. Geological Survey could render invaluable service to the oil industry by addressing themselves to the task of providing an accurate, comprehensive system of nomenclature for general adoption.

### The Magnetic Concentration of Low-Grade Iron Ores

Discussion of the paper of S. NORTON and S. LEFEVRE, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 149 to 169.

S. LE FEVRE, Forest Glen, N. Y. (communication to the Secretary\*).—F. L. Nason thinks I could not have studied Mr. Witherbee's paper and doubts the arithmetic used, but in his discussion arrives at practically the same conclusion. He says 10 ft. thickness of ore over a square mile should give 27,878,400 tons, allowing 10 cu. ft. per ton of ore. This is correct, but if he reads carefully he will see that this depth of ore was figured as covering one-tenth of the surface, which reduces the figures to those used by Mr. Nason.

I did not say, however, that there were 2,300,000,000 tons of ore in New Jersey, but that applying the same method of computation as used by Mr. Nason, to the area of the iron ore-bearing gneisses in New Jersey, we would get a total of 2,300,000,000.

I did say, "In this area are located 366 magnetite mines that have been worked more or less. These lenses may easily be capable of producing (on an average) 1,000,000 tons each, and there are probably double the number listed not opened up."

Thus my estimate for New Jersey, as far as any was made, would be:

|                               |               |
|-------------------------------|---------------|
| Probable ore.....             | 366,000,000   |
| Possible ore in addition..... | 732,000,000   |
|                               | <hr/>         |
|                               | 1,098,000,000 |

Mr. Nason's final conclusion of his personal estimate was one-half of what he erroneously read to be the estimate given in the paper, or half of 2,300,000,000, which equals 1,150,000,000.

So the two estimates seem to differ less than 5 per cent., and we have the two sister States of New York and New Jersey on an equal footing of productive possibilities in magnetic ore.

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\* Received Apr. 24, 1917.

### **An Investigation on Rock Crushing Made at McGill University**

Discussion of the paper of JOHN W. BELL, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 171 to 176.

JOHN W. BELL (communication to the Secretary\*).—I have to thank Mr. Merrill not only for his very kind remarks, but for the weight of his implied opinion that the work accomplished probably has an important practical aspect. I think there is a distinct although unuttered impression among engineers, that it will be time enough for them to begin to take an interest in the law of crushing when it emerges from the cloud of violent and even vituperative controversy that has hovered over it during the last 6 or 7 years. I have a very confident feeling that as the result of the work done by Mr. Gates, by me and by Messrs. Mitchell, Baily and Cockfield, who so loyally helped me to obtain the necessary data, at least the practical law of crushing has emerged; that it is ready and available for use by our fellow engineers, and that it will help them to unravel a number of the many practical problems they are constantly confronted with.

In regard to Mr. Gates' discussion, I would indeed like to believe that McGill could claim, as a minor achievement, the credit which Mr. Gates so generously bestows of having established the "law" of crushing.

I think that more work would be required to establish this claim in the strict sense of the word, as there is a fairly large gap between the coarse and fine crushing results. I am not worried about that gap, because the reason for it is quite clear to me; at the same time I think it will be well to bridge it by a series of tests, and I am bound to admit that the testing machine is the most promising machine right at hand to work with, although whether the minute amounts of power required to crush small quantities of rock can be so accurately integrated as to insure high accuracy in the final result is still a question in my mind. If we had an Amsler-Laffon machine I would certainly try it.

Mr. Gates' remarks about choke crushing are extremely interesting. He compels me to wonder, too. It is a new and probably a very important factor to be considered in this connection.

I am in perfect accord with Mr. Gates about the desirability of the determination of crushing constants, because when this is accomplished the efficiencies (not relative mechanical, but real efficiencies) of different types of rock-crushing plants in any part of the world will then be comparable. For this reason it would seem desirable that the American Institute Committee should not only consider ways and means for accom-

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\* Received Apr. 30, 1917.

plishing this (and the method adopted is immaterial provided it is the best method), but that they should take the matter up with some of the other large societies in and outside of America, to the end of adopting a standard method for determining rock-crushing constants, the efficiency of a crusher, and a standard series of screens.

### The Function of Alumina in Slags

Discussion of the paper of CARL HENRICH, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 2081 to 2086.

CARL HENRICH (communication to the Secretary\*).—I have read with much interest the discussions of my paper by Messrs. A. S. Dwight, E. P. Mathewson, Wm. B. Boggs, Jos. W. Richards, and W. C. Smith.

Mr. Dwight's acknowledgment that he always—in lead smelting—obtained better results by figuring  $\text{Al}_2\text{O}_3$  as an acid in his slags, I may take as a confirmation of my views, notwithstanding his later conversion by the late T. S. Austin to the latter's empirical view and way of neglecting the consideration of  $\text{Al}_2\text{O}_3$  altogether in the composition of slags—letting the slags "dissolve" in  $\text{Al}_2\text{O}_3$ . I have a strong suspicion that Mr. Austin, in a large custom smelting plant at El Paso, has been able to keep the alumina percentage of his slags sufficiently low, not to give him much trouble, although I have also a suspicion that his furnace foreman may have had a somewhat differing opinion of the practical working of the "dissolving" theory, whenever its applicability was tested to its limits. That Austin should have conceived the notion that the Ca of  $\text{CaF}_2$  would enter into a slag as  $\text{CaO}$ , seems hardly in accord with the recognized ability of that late metallurgist. It is a well-recognized fact that even small amounts of  $\text{CaF}_2$  will reduce the viscosity of a slag; but I did not think anyone ever had an idea that  $\text{CaF}_2$ , in some mysterious way, parted with its F and coupled itself with O to satisfy a craving for  $\text{CaO}$  on the part of  $\text{SiO}_2$ —least of all in a lead furnace. The "dissolving" theory for  $\text{CaF}_2$  was doubtless appropriate, and an improvement on figuring the Ca of  $\text{CaF}_2$  as  $\text{CaO}$  to satisfy the demand of  $\text{SiO}_2$  for RO bases. However, I cannot understand the rationality of extending this "dissolving" theory to  $\text{Al}_2\text{O}_3$  and to  $\text{ZnO}$ .

It would be interesting to have more specific data from Mr. Mathewson in regard to the actual composition—analyses—of the slags made, when he had these unusually high alumina slags in Colorado. We "old" smelters all know that within certain limits, usually found out by rather tough experience, we could safely "disregard"  $\text{Al}_2\text{O}_3$ . In fact, I believe, we never really analyzed for alumina, except in slag analyses, when we

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\* Received Apr. 20, 1917.

got it as an incidental to the Fe determination. Specific analyses, furnished by Mr. Mathewson, of successful highly aluminous slags, and also of such which were not so successful, might throw more light on the question of the safe limit to which the "dissolving" theory for  $\text{Al}_2\text{O}_3$  in slags may be pushed—and also on what the constitution of such a slag would be, with or without considering  $\text{Al}_2\text{O}_3$  as an essential constituent of its chemical composition.

Mr. Boggs appears to have found that safe limit of the "dissolution" of  $\text{Al}_2\text{O}_3$  to be near 10 per cent. alumina. I might suggest to Mr. Boggs, that the "not normal" slag made by him (about 33 per cent.  $\text{SiO}_2$ ; 9 per cent.  $\text{Al}_2\text{O}_3$ ; 58 per cent.  $\text{FeO}$ ) ceases to be abnormal if you consider it as  $(\text{Fe, Mg, Zn})\text{O} \cdot \text{Al}_2\text{O}_3 + 2(4\text{RO} \cdot 3\text{SiO}_2) - 32.5$  per cent.  $\text{SiO}_2$ ; 9.2 per cent.  $\text{Al}_2\text{O}_3$ ; 58.3 per cent.  $\text{FeO}$ ; *i.e.*, a normal sesquisilicate slag dissolving—or combining with according to my opinion—about 15 to 16 per cent. of a normal magnesia-zinc-iron spinel. Of course, the presence of magnesia and zinc would further predispose the spinel formation, but is not essential thereto.

I agree with Mr. Richards that my paper would better have been entitled: "The Function of Alumina in Iron-Lime Slags." I may plead in extenuation, that the title was not of my selection; also that the paper was originally written merely as a discussion of Mr. Anton Eilers' discussion of another paper on high-lime slags; and that on invitation of some officers of the Institute I prepared a more extended paper on the function of alumina in slags, for which, however, also on request, I substituted my original discussion of Mr. Anton Eilers' discussion. In that unpublished paper I had dealt also with the function of alumina in slags of iron furnaces. I had come to the conclusion, that while  $\text{Al}_2\text{O}_3$  in these slags, as a rule, plays the role of an acid, a scarcity of RO bases ( $\text{CaO}$  mostly) and a preponderance of  $\text{Al}_2\text{O}_3$  may force  $\text{Al}_2\text{O}_3$  into the role of a base, in part.

Both Mr. Richards' and Mr. Smith's experiences with high alumina slags in iron furnaces are, of course, incompatible with assigning to  $\text{Al}_2\text{O}_3$  in a slag an "inert" character, *i.e.*, with the "dissolution" theory of Messrs. Dwight and Mathewson. It would be interesting, to me at least, to get the analyses of these high alumina slags, both the 10 per cent. and 26 per cent. alumina slags mentioned by Mr. Smith, and the regular 30 to 35 per cent. alumina slag spoken of by Mr. Richards. I believe they would confirm my views on the function of alumina in slags.

### Recrystallization after Plastic Deformation—Grain Growth Phenomena in Metals—On Grain Growth

Discussion of the papers of HENRY M. HOWE and ZAY JEFFRIES, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 118, pp. 1851 to 1860, *Bulletin* No. 119, pp. 2063 to 2073, and *Bulletin* No. 120, pp. 2111 to 2117.

ZAY JEFFRIES (communication to the Secretary\*).—Having seen Mr. Ruder's micrographs of electrolytic iron, I am of the opinion that the tentative explanation offered verbally is correct. Mr. Ruder has informed me that both of the samples were cooled in the furnace. The structures as shown in Figs. *B* and *C* resulted from the last cooling through the temperature range near  $900^{\circ}\text{C}$ . Assuming that the rate of cooling through this range was the same in both cases, the differences in structure of the gamma iron in the two samples must have brought about, in some way, the variations in grain size and grain shape shown in Mr. Ruder's micrographs.

Even though there is practically no temperature gradient in muffle-furnace treatment during the "soaking" period, during both heating and cooling, there is a temperature gradient.

Fig. *B* should normally have been fine-grained prior to the last cooling, due to the comparatively low maximum temperature to which it was heated ( $1,000^{\circ}\text{C}$ .). On cooling through the allotropic change point at  $900^{\circ}$ , these small gamma grains, in conjunction with the temperature gradient, should form an ideal condition for germination. Germinant grains should form near the surface of the sample (the colder portions) and the small gamma grains would be progressively absorbed as they changed into non-gamma iron.

The elongated grains in Fig. *B*, converging toward the axis of the piece of iron, indicate that germinant grains first formed near the surface and advanced their boundaries toward the interior of the sample.

The gamma grains in Fig. *C*, however, should have been large, prior to the last cooling, due to the high temperature treatment ( $1,250$  to  $1,300^{\circ}\text{C}$ .). In other words, to use one of Prof. Howe's expressions, the crystalline organization should be very complete and the change from gamma to non-gamma iron on cooling would take place reluctantly. This resistance to change from gamma to non-gamma iron would probably be greater in the interior of the large grains than at their boundaries. Non-gamma iron centers or nuclei would probably form at a great many places, at the grain boundaries, thus masking the influence, to a large

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\* Received Apr. 27, 1917.

extent, of the temperature gradient. Apparently the effect of the temperature gradient has not been entirely masked in Fig. C, because some of the grains are elongated.

This hypothesis calls for a greater resistance to change from gamma to non-gamma iron, due to its crystalline organization, than the force tending to bring about the allotropic change, due to the interior of a grain being slightly colder than that portion of the boundary nearest the

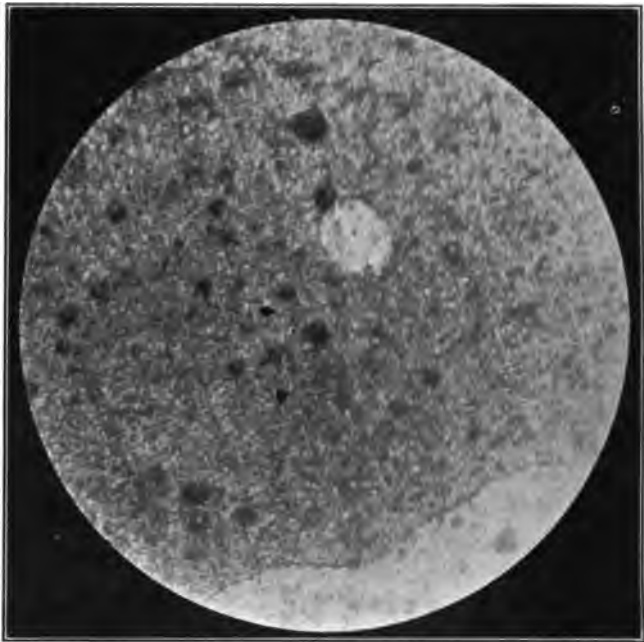


FIG. D.

(hotter) axis of the sample. Small grains would function under these conditions as inactive grains and would be completely absorbed by germinant grains, while under the same conditions, large gamma grains would tend to partition into several non-gamma grains.

Regarding the point mentioned by Mr. Ruder of small grains isolated within large grains, I am submitting a micrograph of tungsten, showing this condition (Fig. D).

# Dry-Hot Versus Cold-Wet Blast-Furnace Gas Cleaning— Some Suggestions Regarding Construction of Hot-Blast Stoves

A discussion of the papers of LINN BRADLEY, H. D. EGBERT, and W. W. STRONG, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, pp. 209 to 228.

LINN BRADLEY, H. D. EGBERT and W. W. STRONG (communication to the Secretary\*).—In the discussion of the paper, a request was made for a further explanation of the curves given on Chart I.

In preparing the curves given on this chart, a typical blast-furnace top gas having the following composition by weight was assumed:

|                       | Per Cent. |
|-----------------------|-----------|
| CO <sub>2</sub> ..... | 21.00     |
| CO.....               | 24.00     |
| H <sub>2</sub> .....  | 0.25      |
| CH <sub>4</sub> ..... | 0.25      |
| N <sub>2</sub> .....  | 54.50     |

For any given top-gas temperature, Curves A, B, C, D, and E indicate in British thermal units the sensible heat lost in cooling the top gas to 60° F. by wet-cleaning methods. Top-gas temperatures are plotted as abscissæ, sensible heats above 60° F. in British thermal units are plotted as ordinates, and separate curves are drawn for top gases carrying various amounts of moisture.

For any given stove exit-gas temperature, curves A<sup>1</sup>, B<sup>1</sup>, C<sup>1</sup>, D<sup>1</sup>, and E<sup>1</sup>, indicate in British thermal units the sensible heat carried out of the stoves by the moisture which is left in the top gas when it is dry-cleaned and enters the stoves hot. This moisture is removed when the gases are wet-cleaned and the sensible heat which it carries out of the stoves is therefore saved. As before, separate curves are drawn for top gases carrying various amounts of moisture.

The curves in Fig. 1, therefore, indicate graphically how much more sensible heat is conserved by cleaning the top gases by the dry-hot method than is conserved by cleaning the top gases by a cold-wet method and thus removing the greater part of the moisture carried by them. They also show of how little moment it is from a heat conservation standpoint whether or not the moisture carried by the top gases is removed before they enter the stoves. That wet cleaning secures the removal of this moisture and is thus to be recommended from a heat conservation standpoint is an argument often advanced by advocates of wet cleaning. The curves of Fig. 1 show how fallacious such an argument is.

\* Received Apr. 28, 1917.

The following example will explain the use of the chart:

Assume the top gas leaving the furnace to have a temperature of 400° F. and to have a moisture content equivalent to 25 gr. of water per cubic foot, calculated at 32° F. Now assume this gas to be cooled to 60° F. before entering the stoves and to carry 5 gr. of moisture per cubic foot, calculated at 32° F. at that temperature. Then from Curve *B* on Fig. 1 it will be seen that the sensible heat lost is approximately 95 B.t.u.

Now assume the gas to enter the stoves at 400° F. and to carry the original amount of moisture. Also assume that the gas leaves the stove at a temperature of 400° F. in both of the above cases. Then from Curve *B-1*, Fig. 1, it will be seen that the sensible heat carried out by the moisture in the gases in excess of 5 gr. per cubic foot, calculated at 32° F. is approximately 5.0 B.t.u. Thus the difference ( $95 - 5 = 90$  B.t.u.) represents the conservation of sensible heat per pound of the gas when the top gas is dry-cleaned as compared to when it is wet-cleaned. Even should the gases carry 100 gr. of water per cubic foot calculated at 32° F. having a temperature of 200° F. at the burner and the exit from the stoves be as before, 400° F., there would still be a conservation of sensible heat obtainable by dry cleaning. This latter case obviously represents an extreme condition.

As has been brought out in the discussion of the paper, there are two factors to be taken into account when considering heat economy in connection with blast-furnace gas cleaning. There is, first of all, the question of heat saving, and then that of getting the heat where it is needed so that an increase in the temperature of the blast can be obtained. In addition, Mr. Wysor brings out an important point in his discussion of the use of unwashed gas; *i.e.*, that the relative cleanness of the gas produced by different systems of cleaning is of great importance, inasmuch as the cleaner the gas the less will be the destruction of the brickwork and the blocking of checker openings with the attendant loss in stove efficiency. In brief, an efficient hot-dry cleaning method preserves the efficiency of the stoves, saves a large amount of sensible heat which is lost in wet-cleaning operations and enables the heat which is thus conserved to be utilized where it is most needed; *i.e.*, in raising the temperature of the blast.



### Potash as a Byproduct from the Blast Furnace

Discussion of the paper of R. J. WYSOR, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 1 to 32.

J. S. UNGER, Pittsburgh, Pa. (communication to the Secretary\*).—In discussing Mr. WYSOR's paper, I said that we were investigating the deposits around furnaces working on certain ferro alloys.

Since the meeting, we have made a number of investigations, and have found that the deposits in the second and third passes of the hot-blast stoves, and near the stack at the boilers of furnaces working on spiegel and ferromanganese, contain about 15 per cent. total potash, of which 10 per cent. is water-soluble. The potash occurs mostly as potassium sulphate, a very small amount being present as potassium carbonate.

### The Significance of Manganese in American Steel Metallurgy

Discussion of the paper of F. H. WILLCOX, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 199 to 207.

EDMUND NEWTON, Minneapolis, Minn. (communication to the Secretary†).—In discussing the question of the deoxidation of steel and material suitable for performing this function at the end of the heat, it would seem logical also to include the question of prevention of oxidation in the steel or the formation of iron oxide,  $\text{FeO}$ . I refer to the use of manganese in the basic open-hearth and basic Bessemer processes.

The advantages of high manganese in pig iron for these processes have evidently been clearly understood in Germany for a number of years, although little has been written in regard to the exact manner in which the benefits are obtained. In this country, while we make no steel by the basic Bessemer process, our basic open-hearth process is becoming more and more important. Until 10 years ago, in this country, manganese in the pig iron for this process was considered of little value and by many writers it was spoken of as a disadvantage. During the past 10 years, however, in certain parts of the United States, higher contents of manganese in basic pig have been used with considerable success, although there is apparently nothing in our technical literature to show the reasons for this trend. A German book, "The Basic Open-Hearth Steel Process" by Carl Dichmann does, however, offer an explanation.

Dichmann says that if the charge for either of the above-mentioned processes contains 3 per cent. of manganese or above, satisfactory steel

\* Received Apr. 28, 1917.

† Received, May 1, 1917.

can be made without further addition of ferromanganese or other deoxidizers at the end of the heat. The steel is not red-short and works well. He says that manganese tends to prevent the formation of iron oxide, the manganese of the metal being oxidized by free FeO in the slag or the bath, and the oxide of manganese combines with silica and lime to form slag. This goes on until equilibrium is established. The presence of considerable manganese in the metal at the end of the heat indicates that little or no FeO is present in the steel. It then only remains to add sufficient carbon to bring the steel to the desired content.

Dichmann's conclusions possibly suggest how the present shortage of high-grade manganese ores and ferromanganese in Germany is being met. There is considerable manganiferous iron ore available in Germany which can be used to increase the manganese content of the pig iron for these processes. In the United States we have no ore suitable for making basic Bessemer steel, but our basic open-hearth process is rapidly increasing. We produce a very small quantity of high-grade manganese ore, although there are available in the Cuyuna Range in Minnesota and in other districts considerable tonnages of manganiferous iron ore, which could be used for the same purpose as ferromanganese made from high-grade manganese ore.

Mr. Willcox raises the question of the Bureau of Mines undertaking a general investigation of substitute deoxidizers for manganese, and I would suggest that they study the various phases of the above-mentioned process. There is the possibility of using this method to a great extent in time of national extremity and also the possibility of its competing successfully with high-grade ferromanganese in ordinary times. It might then be possible to evaluate the manganiferous iron ores against the saving in high-grade ferromanganese. It would, however, seem that such an investigation would be of considerable value and interest to the steel industry at the present time.

### Roll Scale as a Factor in the Bessemer Process

Discussion of the paper of A. PATTON and F. N. SPELLER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, pp. 139 to 148.

F. N. SPELLER (communication to the Secretary\*).—It would certainly be interesting to know more about the heat balance in this practice, but there seems to be no doubt that the net result is a greater absorption of heat where roll scale is used instead of solid metal scrap. In the experimental heats referred to, about an equal weight of pig iron or steel scrap was replaced by roll scale. This caused a reduction of about one-third in the time of blow compared with a heat of the same

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\* Received Apr. 19, 1917.

metal blown under the same conditions using solid scrap but no roll scale. Notwithstanding the fact that there was considerable reduction of radiation losses when roll scale was used, the finishing temperature was observed to be practically the same in each case. This is probably due in a large measure to the fact that when roll scale is added it is not necessary to oxidize  $1\frac{1}{4}$  to  $1\frac{1}{2}$  per cent. of iron in the charge to satisfy the slag. The shortening of the silicon blow is no doubt due not only to the reduction of a large portion of the roll scale by silicon but also to the presence of a surplus of oxide of iron in the bath available for the formation of silicates.

### Temperature Measurements in Bessemer and Open-Hearth Practice

Discussion of the paper of GEORGE K. BURGESS, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, pp. 293 to 306.

HENRY M. HOWE, Bedford Hills, N. Y. (communication to the Secretary\*).—Dr. Burgess would add to the value of this important paper if he would give us the carbon content of the steel made in the several charges, or, if this cannot be done, if he would indicate whether, as might be inferred from his paper, the teeming temperature is approximately constant and independent of the carbon content of the steel, or whether as would be expected, it varies with the content and is materially higher for low- than for high-carbon steel. If he could add to Table 3 the time of the several tests, that would be welcome.

His Table 3 leads one to infer that, once the bath is melted somewhat more than flat, the temperature of the roof is nearly constant and is approximately the highest compatible with reasonable longevity. That is to say, the metal is heated by means of a roof and flame at nearly constant temperature.

The temperature of the observed upper surface of the slag should reflect that of the roof approximately, and in Table 3 this seems to be true, the temperature of both rising toward the middle of the record, and again falling. The slag as the medium through which the heat passes from roof to metal, should be intermediate in temperature between the two, as it is here.

But the temperature of the metal ought to rise continuously, unless when cold additions are charged, and this too seems to be true, allowing for slight observational errors, the total rise from the first to the last spoon-test being  $89^{\circ}$ .

The temperature drop of from  $85^{\circ}$  to  $135^{\circ}$  from the last spoon-test to the teeming is surprisingly great, suggesting that the spoon-tests represent the metal immediately beneath the slag, and therefore appre-

\* Received Apr. 17, 1917.

ciably hotter than the mean of the bath. This is in general harmony with the relative temperatures of slag in furnace, slag stream, and metal stream.

In three of the cases on page 300 the slag stream is materially cooler than the slag in the furnace, and the metal cooler yet, from  $28^{\circ}$  to  $41^{\circ}$  cooler than the slag in furnace. In the fourth case the slag in furnace deviates from this order, and is  $49^{\circ}$  cooler than the slag stream and even  $18^{\circ}$  cooler than the metal stream, indicating that the furnace lay for some time without gas just before tapping, so that the visible upper surface of the slag cooled down to well below the temperature of the underlying slag layers, and even to below that of the metal.

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### Erosion of Guns—The Hardening of the Surface

Discussion of the paper of HENRY FAY, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 120, December, 1916, pp. 2237 to 2251.

HENRY FAY (communication to the Secretary\*).—In reply to the discussion of my paper, I wish to take this opportunity to express my appreciation to the Chairman and the various gentlemen who have taken part. I regret that others have not offered criticism, for honest differences of opinion often lead to the truth, and suggest new lines of work.

An experiment has been made which bears on Prof. Sauveur's discussion and which helps to emphasize the conclusion that cementation is not a factor in the hardening of the surface. Dr. Langenberg, of the Watertown Arsenal, has heated for me, *in vacuo* at  $1,000^{\circ}$  C., a piece of the metal containing a hard surface. The microstructure shows uniform distribution of carbon throughout the metal, showing that no carbon has been added to the surface and heating *in vacuo* has prevented any oxidation of the surface.

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### Notes on the Heat Treatment of High-Speed Steel Tools

Discussion of the paper of A. E. BELLIS and T. W. HARDY, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 61 to 68.

HENRY M. HOWE, Bedford Hills, N. Y. (communication to the Secretary†).—The authors' valuable results as to the effects of the air-hardening temperature on high-speed steel may be summed up thus:

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\* Received Apr. 24, 1917.

† Received Apr. 29, 1917.

*Influence of Rising Air-Hardening Temperature on High-Speed Steel*

| Air-hardening temperature | Microstructure       |                                   |                     | Cutting Properties       |
|---------------------------|----------------------|-----------------------------------|---------------------|--------------------------|
|                           | Austenite Grains     | Grain Boundaries                  | Free Carbide        |                          |
| Too low                   | Small or undeveloped | Narrow                            | Maximum             | Too soft                 |
| Intermediate, correct     | Well-developed       | Intermediate                      | Little or even none | Best                     |
| Too high                  | Coarse               | Wide, even with three-rayed stars | Minimum             | Weak, brittle, crumbling |

Here the suggestion is that as the air-hardening temperature rises it causes two prominent effects, one helpful and the other harmful, the increase of the former exceeding that of the latter up to a certain point, at which the steel is most useful, and then in turn being outrun by it. The first is the progressive dissolving of the free carbide in the austenite, increasing the stability of the red-hard state. The second is the progressive coarsening of the grains, and thickening of the intergranular cement, developing at last into spandril-like three-rayed stars of the 1,288° specimen (2,350° F.) of Plate *D*, thus giving clear proof of incipient fusion, that is of crossing the solidus. This causes intergranular weakness. The individual grains may have great red-hardness, but they do not cohere. They approach the condition of a bag of emery grains.

May we not sum this up by saying that we should, by raising the temperature, increase the red-hardening effect of dissolving the carbide, as far as we can without causing undue intergranular brittleness through the simultaneous grain coarsening, and the thickening of the intergranular cement?

A. E. BELLIS (communication to the Secretary\*).—The valuable discussion of our paper has brought out many interesting points regarding the heat treatment of high-speed steel. In reviewing it we feel that we should have been more emphatic in our general conclusion that the best performance results from tools having a fine-grained austenitic structure obtained in the shortest time possible. In cases of cutters where the surface must be preserved, the reducing atmosphere of a gas or electric furnace is more reliable than packing or the use of salt baths. The tools should be quenched the instant they reach the desired temperature. The effect of time on the coarsening of the grain is illustrated by two microphotographs presented herewith.

Cutters that require great strength, even at the expense of hardness, such as small profiling tools and saws, should be given the lowest possible hardening temperature and finest possible austenitic grain. Cutters that have given phenomenal life, on inspection after failure, have shown an infinitesimally small network, which was recognized as austenite only by the brightness of the specimen after etching.

Tools that have been tempered give a gray or black "background" after a short etching, even if the structure is austenitic; the higher the drawing temperature, and the longer the etching, the darker the "background." It is especially desirable, therefore, to use the method of

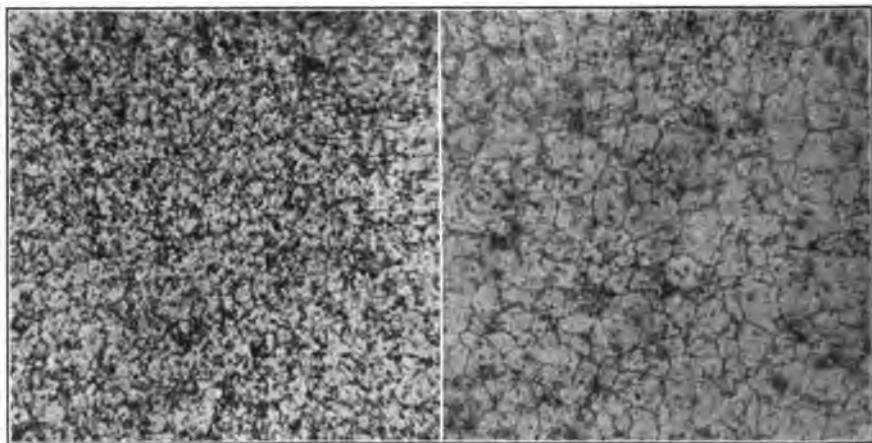


FIG. A.—IN FURNACE AT PROPER TEMPERATURE, 1 MIN.  $\times 560$ .

FIG. B.—SAME STEEL AS FIG. A IN FURNACE AT SAME TEMPERATURE, BUT LEFT THERE FOR 15 MIN.  $\times 560$ .

"quantitative etching" advised by Mr. Langenberg in examining tools after the tempering. But when we come to measuring degrees of color and seconds of time we approach the borderland of science and assume the prerogatives of artists in heat treatment.

### Portable Miners' Lamps

Discussion of the paper of EDWIN M. CHANCE, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 235 to 243.

EDWIN M. CHANCE (communication to the Secretary†).—The subject of miners' lamps has long interested me and it would seem that the thought that has been given to it by the Bureau of Mines, and the users

\* Received Apr. 18, 1917.

† Received Apr. 20, 1917.

and the manufacturers of these lamps, is about to bear rich fruit. Not only will the solution of this perplexing problem mean greater safety and comfort for the mine worker, but it will also mean decreased rates of insurance because of this increased safety. There should also be induced greater efficiency of the individual worker, with accompanying greater output per man. Under the present labor conditions and those of the immediate future, this is all-important. The attainment of these results can be looked for in the immediate future with confidence.

### Report of the Secretary of the Committee on Safety and Sanitation

Discussion of the paper of E. MALTBY SHIPP, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 87 to 122.

C. W. GOODALE, Butte, Mont. (member of the committee)—(communication to the Secretary\*).—In the discussion of papers presented at the New York meeting of the Institute in February, 1915, and at the Arizona meeting of September, 1916, I referred to the activities of the Bureau of Safety of the Anaconda Copper Mining Co., and I am now submitting a few thoughts by members of the staff of that Bureau.

TORGUS H. OAAS, Safety Engineer, Butte, Mont.—The report of the Committee on Safety and Sanitation has described different organizations and methods of promoting safety work. The effectiveness of these methods would aid the beginner in his campaign for safety in choosing the kind of organization and methods best adapted for his field. This can be obtained only by a discussion by those who have long promoted this kind of work, and have met with successful results. As the field is new, the different organizations promoting safety work have not established a common basis of rating for their fatal, serious and slight accidents, so that it is now difficult to get a comparative basis from which to judge the merits of the different systems.

C. W. GOODALE.—The U. S. Bureau of Mines adopted, at the beginning, as a basis for its yearly summaries, the number of accidents per 1,000 men employed, and it is now (quoting from its report for 1915, *Technical Paper* 168) making the following "classification of injuries:"

"Many of the States now have compensation laws, and in order to conform with their classifications, the bureau's classification of serious and slight injuries for 1915 is on a 14-day (2-week) basis instead of 20 days, as in previous years. The new classification of injuries includes three types, as follows:

1. Fatal
2. Serious (time lost, more than 14 days)
  - (a) Permanent disability
  - Total
  - Partial

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\* Received Apr. 13, 1917.

## (b) Others

## 3. Slight (time lost, 1 to 14 days, inclusive)

*"Permanent Total Disability"*—Loss of both legs or arms, one leg and one arm, total loss of eyesight, paralysis or other condition permanently incapacitating workman from doing any work of a gainful occupation.

*"Permanent Partial Disability"*—Loss of one foot, leg, hand, eye, one or more fingers, one or more toes, any dislocation where ligaments are severed, or any other injury known in surgery to be permanent partial disability."

For our own purpose, that is, to enable our superintendents and foremen to follow the progress of our safety campaign, and the part which each one takes in the movement, we have adopted the shift or day's work as a basis, considering each shift as a risk, but in order to avoid too many decimals, we show the number of accidents per 10,000 shifts. The difficulty in using as a basis the number of accidents per 1,000 men employed will be seen when attention is called to the fact that on Sundays and holidays work is not entirely suspended, as in the case of a factory, with only watchmen on duty. These days are taken for repairs in shafts and on equipment, and this work requires many men. To arrive at the exact average number of men employed for a year or for any period would therefore call for a good deal of arithmetic.

Referring to the classification of injuries, the Bureau of Safety of the Anaconda Copper Mining Co. has insisted that all accidents, however slight, shall be reported, as it may happen that a trivial injury will develop infection, and it wants a complete history of every accident.

During the year 1916, the accident record covering underground and surface operations was as follows:

| Kind of Accident                          | No. of<br>Accidents | Per Cent. | Accidents per<br>10,000 Shifts |
|---|---------------------|-----------|--------------------------------|
| Fatal <sup>1</sup> .....                  | 46                  | 0.76      | 0.12                           |
| Serious, time lost more than 14 days..... | 301                 | 4.97      | 0.79                           |
| Slight, time lost 1 to 14 days.....       | 2,913               | 48.09     | 7.64                           |
| Trivial, no time lost.....                | 2,797               | 46.18     | 7.34                           |
|   | 6,057               | 100.00    | 15.89                          |

<sup>1</sup> 21 of the 46 fatalities were due to a fire in the Pennsylvania mine.

T. H. Oaas.—Our experience in the mining department of the A. C. M. Co.'s Bureau of Safety has proven that the strictest disciplinary measures must be enforced if accidents are to be reduced. The payment of a cash bonus to the mine foreman having the least number of fatal and serious accidents, in proportion to the shifts worked, has been tried with some measure of success. A set of rules covering all dangerous practices has been established with an attached penalty which must be inflicted impartially on all offenders. This not only affects the miner, but also the



shift boss and foremen who neglect to instruct their men in the necessary precautions before they undertake any work.

You may be sure whatever respect or interest the foreman and shift bosses have for the safety of their workmen is reflected in the manner in which they do their work. The shift boss, or foreman, who says to his men "How's she going?" and does not look for dangerous slabs, poor floors, unguarded chutes, misfires, etc., in the working places, is not the leader of that crew of men, but only an extra timekeeper. That type of shift boss or foreman is the one who is a detriment to the safety work of his company. These men in charge must caution their men about that dangerous slab, or that unguarded chute, and they must see that these unsafe conditions are remedied at once. Should they pass over these conditions, their men will do likewise. Some shift bosses have compelled their men to work under loose roofs and sides, fearing the loss of a car or more of ore for the time taken to secure the loose roof and sides. This is a very common and mistaken practice, for safe methods have proven that they are necessary to efficiency. The A. C. M. Co. has handled this problem by discharging shift bosses and foremen who fail to look after the safety of their men, and obey the safety rules of the company. By insisting on safe practices and appliances, and by strict disciplinary measures backed up by the operating department it has convinced the majority of the men of its earnestness in safety work. During the year 1916 there was a reduction of 35.9 per cent. in the fatal and serious accidents happening at the mines of the company employing about 11,000 men, by comparison with the year 1915. This was due largely to the establishment of a 7-day layoff as a penalty for the first offense of disobeying the safety rules, 2 weeks layoff for the second offense, and discharge for the third offense. This method seems logically correct, and has been successful both in reducing accidents, and securing the cooperation of some of the old-timers who were the hardest men to convince that they should be more careful in their work.

A monthly bulletin, *The Anode*, is printed and distributed to all employees. One feature of this bulletin is the publication of the accident records of the different mines and shops, and this stimulates interest on the part of both foremen and men. Two prizes of \$15 each are offered monthly, for the two best articles on some feature of safety work connected with the operations upon which the author is engaged. One of these prizes is awarded to an employee of the metal mining department, and the other to an employee of the reduction works, or other departments outside of Butte. It also publishes pictures, taken in the mines, of bad and good practices, and these guide the beginner into the right channel. Bulletin boards at the mines covered with bulletins of local interest also tend to keep before the men that important watchword "safety first"; which slogan the majority of men are now approving,

realizing that it decreases the misery and hardships caused by needless accidents.

To further promote the interest of the shift bosses and foremen, there is a meeting each month of each of the four mine safety organizations composed of the shift bosses and foremen of the various mines. These meetings are presided over by the general superintendent of mines, who reads and discusses every accident report of the previous month concerning accidents which happened to the men working under the shift bosses and foremen present. The reports of the two mine safety inspectors as to the condition of the working places in the mines visited by them is also read. New safety devices and infringements of safety rules are discussed. This gradually awakens the interest of the mine foremen from a lukewarm condition until it causes them to become most zealous advocates of the safety-first movement.

Sanitation in the mines is promoted. All mines are provided with sanitary toilet cars, sanitary drinking tanks, and some of the mines have installed sanitary drinking fountains. Stopes, cross-cuts, and drifts are provided with water so that men may wet down the dust.

Instruction in first-aid and mine rescue work is an added effort to bring home to the men the fact that the company is seeking their fullest coöperation in safety work. All employees who are desirous of learning first-aid and mine rescue work are taught by an able instructor who has charge of such work. Two rescue stations are located centrally so as to accommodate the mines for any emergency as quickly as possible. There are two modern automobile ambulances to carry injured men from the mines to the hospitals.

The safe and fireproof installation of electric fans, motors, trolley wires, electric wires, and power lines, is a further factor in safety to which too much attention cannot be paid. They are a constant source of dangerous fires. Experience has shown that one fire may cause more fatalities than will result from any other cause in a whole year.

The following additional signs are suggested for shaft stations:

"Danger. Do not stand near the shaft."

"Danger. Hands Off." (Placed at each shaft bell.)

"Call Bell" or "Buzzer" "1,700, 5-2." •

"All hands are requested to keep this plant safe and clean."

"Men are prohibited from eating their lunches on this station."

Added signs for mine workings:

"Danger. Do not use this manway."

"Do not dump here."

"Men working on wire. Do not close switch."

"All lighted candles must be placed in candle sticks or sconces."

The following additional safety rules are suggested, and the reasons given after some of the rules:

1. A miner's first duty on entering his working place shall be to clean and fix his floors so as to give a safe retreat, and then bar down the roof and sides before working under them.

2. In all manways that are vertical, or only slightly inclined, platforms should be constructed every 30 ft., and where these platforms occur, substantial guard rails should exist between them and the timber slide.

3. All chutes and working raises shall be covered with 10 by 10 grizzlies. Openings between grizzlies must not be more than 10 in., and they must be properly blocked. They shall also be guarded with railings.

4. At all stations of hoisting compartments where skips are used the stations must be equipped with gates to keep the falling rocks, caused by the overloading of skips, from flying into the station.

5. When explosives are brought to the shaft on the surface to be lowered into the mine, hoisting rock by skips must cease at once.

6. Powder must be lowered in powder cars that are covered to prevent falling rocks in the shaft from striking the powder.

7. When powder is taken off at the station all power to the trolley lines must be shut off and powder trammed to the magazine by hand.

8. Powder magazines must be kept clean.

9. A box, boat, or steel car especially built for hoisting and lowering steel must be used.

10. In side dumping a mine car, a support must be used to prevent the car from tipping backward.

Many accidents, to the extent of broken limbs, caused the enforcement of this rule.

11. When lowering and hoisting timbers in manways, a chain equipped with a hook to be driven into the timber and keep the chain from slipping, must be used attached to the rope.

12. All trolley wires must be boxed, no matter how high above the rail. The boxing must extend 3 in. below the wire.

14. Mine cars, whether loaded or empty, moved by a motor, must at all times be coupled so that the cars are under direct control of the motor man.

15. All timbered sets must be backlagged next to the roof.

16. Cages and skips must be inspected daily, and at the time of inspection all cages in the several compartments must be at the surface.

17. In rill stopes safety chains must be used instead of ropes, because the ropes are quickly cut to pieces by rolling rocks and blasting.

18. Men are forbidden to take short cuts through dangerous workings.

19. Men are forbidden to take wine, beer, or other intoxicating liquors underground.

20. Strangers, and visitors, shall not be allowed underground unless accompanied by some employee designated to accompany them.

21. When timber is being lowered into the mine on multiple-deck cages, and successive deck loads are to be unloaded at different levels, the timber for top levels should be unloaded first, and the remaining decks in consecutive order on levels below.

22. The use of carbide lamps is recommended for spitting fuses in wet drifts, and in shaft sinking.

23. All motor seats must be fixed on the motor car, and they must be equipped with back rests.

There is generally not much clearance underground between the motor and the ground or timber, and there is danger of the motorman sitting out over the motor seat and being squeezed between the motor and mine timber when the seat is not equipped with back rest.

24. "Firebugs" shall be sent through the working places after each shift.

25. Traveling about unlighted stopes or drifts without a light is prohibited. When underground, men should always carry matches and candle, or lamp.

Some of these rules were adopted after a fatal or serious accident had occurred because of the dangerous practices we now prohibit, although before the occurrence of the accident such practices were not considered dangerous. Past experience has thus proven the necessity for the adoption and strict enforcement of the rules.

A. S. RICHARDSON, Safety Engineer, Butte, Mont.—In the report, mention is made of the use of printed books of safety rules for distribution to workmen to inform them of the regulations that have been made to prevent accidents, and by which they are to be governed. In some cases men are required to pass an examination to show their knowledge of the contents of these books of safety rules, but in other cases no such effort is made. At a large mine, or plant, the task of undertaking this examination of new men entails a lot of work, and since it is likely that the information, once gained, is quickly forgotten, it is doubtful whether the purpose for which the rule books are distributed is not better attained by the use of bulletin boards, or other publicity measures, which renew interest periodically.

The establishment of any safety rule should also be accompanied by the establishment of a definite penalty to be imposed in case of violation. Making safety rules that can be broken with impunity is worse than useless, yet none the less common.

Bulletin boards, to be effective, must be placed where men have time to stop and look at them, and must present information that is constantly new and interesting. Bulletin boards presenting a large quantity of stale material, and placed near a timekeeper's office, past which the men always pass when in a hurry, have little value. Bulletin boards located

near a drinking fountain, or some similar place, where men may be when they have a minute to spare, and which present information that they can look over in a short space of time, have great value, particularly if the information has local interest, and is changed at reasonably short intervals.

Where foreigners are employed, it is desirable that printed bulletins should appear in their native language. Pictures are, however, universally and quickly intelligible, so that they are decidedly the most valuable material to use.

Attention may be called to the importance of showing by comparative data what each shop, mine, or department is doing, relatively, in the work of reducing accidents. Such comparisons, when justly made, do much to stimulate a spirit of emulation that is of the greatest value.

Unfortunately for the success of the safety-first movement, there is as yet no universally accepted rule for calculating accident rates.

The solution of the problem of arriving at a universally applicable method of determining accident rates is, we believe, much simplified by getting down to fundamentals. Primarily there are two things to be considered: the accident, and the risk of accident. There is as yet no common basis for classifying accidents, but a classification based upon "time lost" and "disability" is being rapidly accepted. The risk of accident under given conditions seems most exactly to be measured by the time a man is employed under those conditions, and so it would only be logical to adopt some unit of time of employment, or multiple of units, as the basis upon which to calculate accident rates. In the work of the A. C. M. Co.'s Bureau of Safety, the number of accidents of each class per 10,000 shifts, or 10,000 8-hr. working days, are used as accident rates. There have been no objections to the use of this basis of calculation from any of the operating officials; the information has been readily obtainable; and we confess that efforts to supply others with information as to accident rates based upon the average number of men employed were not satisfactory.

As a part of safety work related to metal mines, the provision of sanitary toilet cars, and properly protected cars for drinking water, for use underground, is necessary. In the Butte mines the special steel toilet cars that are used are equipped with hinged lids which keep the cars tightly sealed when not in use, and the cars themselves are partly filled with a disinfecting fluid. At the end of each shift all toilet cars are taken to the surface, emptied through a valve in the bottom of the car, sluiced out, and prepared for further use. Iced drinking water is sent into the mines in closed cars which are securely locked so that nobody can take out ice, or touch the water, and which have the taps protected so that nobody can touch them with their hands or mouths. Open kegs

for use in the stopes are being taken out of service as rapidly as possible, and only closed tanks will be used.

Mine ventilation, and the prevention of dust in mine air, are matters whose importance does not seem to have been fully recognized by the metal mining industry in this country. In deep mines, the hot, humid, air is debilitating, affecting the general health of the miner, decreasing his desire and ability to protect himself from injury by accidents, and greatly reducing the amount of work he is able to do. "Miner's Con" is well recognized, but systematic efforts to do away with the conditions that induce it have been much neglected. A far-sighted and humane policy in dealing with these matters is much needed, and undoubtedly all effort made would be well repaid.

In connection with mine ventilation, stress should be laid upon the very great importance of making all housings and surroundings of mine ventilation fans absolutely fireproof. Electrical connections and apparatus used to drive the fans are notorious incendiary agencies, particularly when under the control of unskilled or careless operators. Probably no one thing can endanger so many lives in a metal mine as a fire starting near a ventilation fan which may quickly blow the smoke through the mine workings.

First-aid and mine rescue work is rapidly becoming an important feature of the safety-first campaign, but at many metal mines the work is still limited to the training and practice of a first-aid or mine rescue team which shall represent the mine at contests, and which hardly returns a maximum of service for the work and expense involved. To provide prompt first-aid assistance for all men who may be injured in a mine which has 50 or more scattered working places is a difficult problem. It cannot be solved merely by training a first-aid team of six men, for it is more than likely that but few of these men will be working when the accident occurs, and that it will take some time to find them. Also, it is quite likely that if the first-aid man has not been training for some time he will be badly out of practice. Effective service requires systematic training and practice of a sufficiently large number of men, and properly planned arrangements so that both first-aid men and material may be quickly available at any locality in the mine.

In conclusion, emphasis should be laid upon the one thing that more than all else is necessary for the success of a safety-first campaign; namely, the necessity for the active support of the management. It is, we believe, an unfortunate fact that some companies have safety-first departments that were created as a sop to progressive opinion, but that these departments accomplish little or nothing toward the purpose for which they were created merely because they are not supported in their efforts by those active in the management or control of operations. A reduction in the accident rate at any mine or plant can be accomplished

only as a result of changes, either in the care used by the men in doing the work, or in the equipment and methods used. Necessarily this means changes in the operating departments, and, generally, outside influences trying to produce such changes will not be welcomed if they can be ignored. The accomplishment of safety-first measures, therefore, demands such active support by the management as will compel sincere coöperation by the operating departments, or else delegation of full authority to the safety department. For complete success there can be no halfway measures.

**JAMES L. PRICE, Great Falls, Mont.**—The part a foreman plays in a safety campaign, and how to secure his sustained interest:

It is generally agreed that there is no set rule by means of which best results can be obtained in an accident-prevention campaign. Every industry, each individual plant, must learn to solve the various safety problems that present themselves in some way best adapted to the particular organization in question. Some claim to have obtained good results through the functions of so-called safety committees; others achieve gratifying results in accident-prevention work by placing the onus on their foremen; still others rely upon the combination of both to minimize industrial hazards, while the rest are continually devising methods altogether their own.

After 3 years of intensive study of the subject, we feel that in our particular work, the metallurgical branch of zinc and copper mining, placing the responsibility upon the foreman appears to be the most efficacious method in accident-prevention work. Undoubtedly, every move and step that has for its aim accident prevention, if not too far-fetched, or absurd, is bound to do some good, even if it does seem to the onlooker "muddling" and to a large extent a waste of effort and time. Since we all, however, are desirous of obtaining in our accident prevention campaign a maximum of efficiency of the lasting kind, we must frown upon any and all methods that savor of the haphazard and the slipshod. Sooner or later such methods are sure to prove harmful for the very simple reason that we acquire a false sense of security, and when disaster of any magnitude overtakes us, we are not prepared to cope with it.

At the fifth Congress of the National Safety Council, A. H. Young, then Supervisor of Labor and Safety of the Illinois Steel Co. at South Chicago, and now Director of The American Museum of Safety, presented a brief paper in which he contended that the surest method to prevent accidents is to hold the foreman in whose department accidents happen to strict accountability. Through an unfortunate association of ideas, most of the 2,000 members of the Council present took the expression of "strict accountability" in the same spirit in which the Romans

must have taken Antony's: "Brutus is an honorable man." . . . Yet the more thought one devotes to safety as applicable to a plant, the stronger grows the belief that the solution of the problem is held by the foreman and by no one else. To be sure, as adjuncts in the safety campaign, it does no harm to have a workmen's safety organization; safety committees made up of the rank and file have a certain, if questionable, value, but to prevent the most accidents with the least expenditure of effort and money you have to come to your foreman, to the man who hires and fires; every other means must either coördinate with this method or become subordinate to it.

From the foreman's point of view, however, such reasoning is unjust. He, the foreman, does not want to be held responsible for the accidents that may happen in his department. He has, for so many years, mentally evaded the responsibility that moral and physical evasion comes as if it were second nature to him. Any ill-considered, tactless attempt on the part of the employer to change this attitude of the foreman is bound to bring about friction, misunderstanding, frequently quite silly, in some cases open rebellion. For so many years, the foreman has been in the habit of placing the responsibility on the man who met with the accident, that he cannot, will not, see things now in a different light.

We are all familiar with the report that the average foreman hands in to the management after an accident in his department. In 90 cases out of 100, the cause of the accident, according to the theory advanced by the foreman, is: "His own (meaning victim's) carelessness," "No one to blame but himself," "Purely accidental," and so on. The foreman makes no mention, probably the idea never occurs to him, as to whether anything might have been wrong with the method that was used at the time the accident happened, or who is responsible for the method in vogue. When cornered and pinned down he will cut the knot with: "he (the man) should have been more careful." In this manner he clears his conscience and is perfectly satisfied that he has done his duty toward his employer and to his subordinates.

Now let us see if we cannot analyze to some extent the reasons that prompt us to assume an entirely different line of thought. Let us look into the reasons that justify us in placing the responsibility on the foreman, and furthermore, why he should bear this responsibility.

First, let us put ourselves in the place of the foreman and let us reason as we want him to reason:

"I am, then, the foreman. An accident has happened to one of my men. Within 10 min. after the accident occurred there will be some one out to investigate the causes and to properly fix the blame. Witnesses will be summoned and questioned, the details of the operation will be gone into, the condition that prevailed at the time of the accident will be analyzed. I know that if a possible remedy, or means of prevention,



suggests itself to the investigator, such will be embodied in the report to the management. I know it will be useless for me to say that the man was hurt because of his carelessness, or through the carelessness of a fellow workman. I know that the management will call me in and begin to ask questions which, I somehow feel, reflect upon my ability to handle men. I know that if the report to the management proves that the accident could have been prevented, and would have been prevented by the exercise of closer supervision, or by giving the man more explicit instruction, I am due for a week's vacation without pay. I know that if accidents, even such as I honestly and sincerely consider to be unavoidable, continue to take place in my department, the management will become prejudiced against me, and will begin, before long, to look around for another foreman. In other words, I know that if I care to remain on the job, I shall have to pay more attention to the way my men are doing things. Then, of course, there is another way of looking at this safety-first campaign. At first I thought that it was a fad. I felt that while not altogether a joke, the interest that the management displayed was more or less superficial and perfunctory. I had an idea that if ever things should come to a showdown—production first or safety first—safety first would come in last. I can see now how mistaken I was. Of recent months I have been told explicitly, insistently, firmly, that the safety-first idea is not a side issue and must not be considered as such either by myself or my men. I am almost beginning to suspect that from the management's point of view safety first comes ahead of production. It is an unheard of state of affairs. It is incredible, and that is the reason it took me so long to grasp the significance of this new phase in the life of the plant. But if such is the case, then there is a bright side to it. If I make it a point to prevent accidents in my department, I think I can accomplish it just as well as I did other things. Then, because I am painstaking, because I tell my men what I want done and how I want it done, because when there is any doubt in my mind as to whether I made myself clear to my men, I show them how to do things safely and efficiently, because I am on the job 60 min. an hour and make every minute count, I actually succeed in keeping accidents in my department down.

"Months go by. No one of my crew gets hurt. No one of my men caused an accident to men in other departments. I feel the management is aware of the work I am doing. The management sees that I have fallen into line and am doing the things other foremen declared were impossible. My monthly check begins to carry with it a little prize, not much to be sure, still it is something tangible, it is proof of recognition. The management thinks well of my efforts and that little extra money is just a sign of appreciation. . . . Who knows, but some day there may be an opening to a somewhat bigger job, with some more pay and still

better prospects. . . . What then am I going to do about accident prevention?

"I know what I am going to do.

"I shall see that each man who is hired by me is thoroughly informed what he is to expect from me if he is caught taking a chance. I shall make it very plain to him that I will not tolerate for an instant in my department any one who disregards safety rules. It will be an open secret among my men that the careless, thoughtless, indifferent man will be gotten rid of at the very first opportunity . . ."

Thus the foreman. Now we place ourselves in the position of the workman who is employed by the foreman whose line of thought we attempted to reproduce:

"I, the workman, know that there are orders out from higher up to cut out accidents at this plant. I know that Bill, John, Bob, Tom, Dick, Steve, have been let out at some time or other, because one did not wear goggles while babitting, the other for throwing a plank from off the roof without first looking to see whether there was some one below, the third for using a mushroomed chisel bar, the fourth for leaving a manhole open and failing to put the guard up, the fifth for coming out to the works under the influence of liquor, the sixth for not reporting a dangerous condition he was aware of . . . I know that if my job is worth keeping I must not jeopardize my foreman's bread and butter, for 'they' put it up to him, and he puts it up squarely to me. I know that if I am ever caught indulging in an unsafe practice I am going to be laid off or discharged. As I think it over, I can be as careful as they want me to."

So much for the workman.

If our analysis is correct, if it is understood by the foreman and by his men that the foreman is looked to for a minimum of accidents just as much as for a maximum of production, it is fairly safe to say that preventable accidents will not happen with dismaying frequency and discouraging persistency.

The question presents itself immediately: How can we induce the foreman to look at the accident-prevention problem from our point of view? How are we to teach him to consider our problem, his problem? How are we to convince the foreman that the matter of accident prevention is up to him? Shall it be through instilling in him a fear of punishment? Or shall we hold out to him the hope of reward? Which of the two offers the stronger incentive? Or perhaps, shall we use as a propelling motive both of these interest-developing stimulants? These are the all-important questions in connection with accident-prevention work that managements are now called upon to decide. When an accident takes place and after having analyzed the surface and underlying causes we can fasten the blame on to the foreman, what are we going to

do about it? Shall we lay the foreman off? Shall we discipline him in some other manner? On the other hand, let us assume that a foreman has no accidents at all, let us say, for the sake of argument, that as far as accidents are concerned this foreman's record is absolutely clear. Are we to pay him a bonus? Or raise his salary? Or mark him for promotion? Or, a judicious application of both theories would work out well. That is to say, combine fear of punishment with hope of reward, or rather hold out to him the contrast between the two. For it would seem that if when a foreman has accidents he is either demoted, or discharged or if, on the other hand, when he takes an interest in safety, eliminates unsafe practices, insists upon the exercise of care, keeps his tools and machinery in good condition, his pay is increased and he stands in line for promotion, the contrast between the two extremes is made stronger, and so is the incentive to make good.

### The Tayeh Iron-Ore Deposits

Discussion of the paper of CHUNG YU WANG, printed in *Bulletin* No. 123, March, 1917, pp. 367 to 373.

THOMAS T. READ, Palmerton, Pa. (communication to the Secretary\*). —It is much to be regretted that in presenting a further discussion of these deposits, first described by myself<sup>1</sup> and later discussed by C. M. Weld,<sup>2</sup> Mr. Wang has not made a more complete presentation of the evidence on which he bases his conclusions as to their origin. Mr. Weld said in his paper that his visit was confined to a few hours, that he saw only one of the several exposures, and that he took away no samples for further study. My own visit was brief, and for another purpose than the study of the geological features of the deposits, so the fact that Mr. Weld described as a hornblende-granite the rock which I described as a "dark gray syenitic rock," and which Mr. Wang identifies as a hornblende-syenite, seems of no great importance, since this rock is generally decomposed along the contact, presumably the only place in which Mr. Weld observed it. Later in the same paper (p. 31), Mr. Weld says that his opinion is that the magnetite of the deposits is primary, but Mr. Wang makes no mention of this in stating his own conclusion to the same effect. When Mr. Weld's paper appeared, I did not discuss it, because, although I considered the evidence that he brought forward to support his belief that the magnetite is primary too slender to support the hypothesis, my own evidence to the contrary was equally tenuous. The opportunities for observation and study open to Mr. Wang must have provided him with

\* Received Apr. 2, 1917.

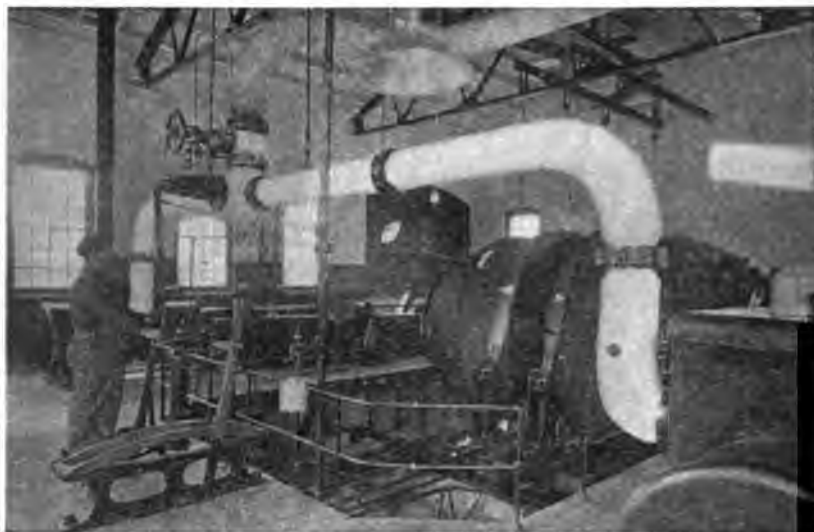
<sup>1</sup> *Trans.* (1912), 43, 29-30.

<sup>2</sup> *Trans.* (1912), 44, 27-37.

abundant evidence as to the origin of the deposits, and it is much to be regretted that he has not seen fit to include it in his paper, instead of making a brief statement of his conclusions.

Before accepting the statement that these deposits are of contact-metamorphic origin, the magnetite being primary, it is proper to inquire how closely they resemble the deposits elsewhere which have been ascribed to contact-metamorphic action. In the Banat of southern Hungary, there are iron deposits which various European geologists are agreed are due to the eruption of syenitic rock through the Tithonian limestone. The iron ore is magnetite, but it is accompanied by garnet, wollastonite, vesuvianite, and other common products of contact metamorphism. In Russia, at Wyssokaia-Gora, in the Nijne-Tagiulsk district, syenite has intruded Devonian limestones. Garnet-epidote rock is abundant, and is generally considered to be the product of the contact metamorphism. The iron deposits of the Island of Elba are invariably accompanied by garnet and epidote. The deposits at Traversella and Brosso, in the Piedmont, Italy, are famous for their garnet crystals. At Dannemora, in Sweden, the limestone shows characteristic garnet. It would seem, therefore, that any complete discussion of the origin of the Tayeh deposit must include an explanation why, to quote Mr. Wang, "hardly any trace of contact minerals can be found." It is also difficult to reconcile the presence of "chalcocite found disseminated in certain portions of the orebodies" with an oxidizing action sufficiently vigorous to convert magnetite into hematite. I do not deny that these deposits may be due to contact action, but the evidence so far presented to support the hypothesis appears to be too incomplete to warrant its unreserved acceptance.

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Other interesting information announced concerning the district is contained in a statement of the variety of mineral products that it is sending out in addition to its production of copper ore. Included are zinc-lead shipments to Kansas City, used in the manufacture of paint; manganese shipped to Chicago; sulphuric acid made from local sulphides, which goes principally to the New Cornelia Copper Co. at Ajo, where it is used in the solution with which that company is recovering the copper from its carbonate ores by the additional application of electric current. In addition, the district is shipping fluxing ores, mainly sulphides, to Sasco, Globe and El Paso smelters in aid of their operations.

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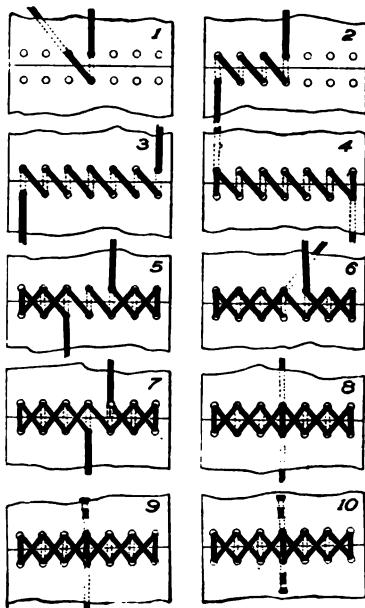
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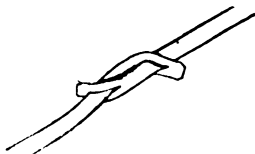
## Two Good Ways



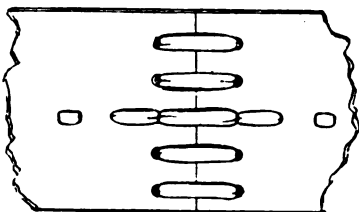
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### Trade Catalogs

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The Allis-Chalmers Mfg. Co. has issued a new bulletin, No. 1632A, on Centrifugal Pumps and Pumping Units, together with a leaflet describing the official test of the Allis-Chalmers 10-in. single stage pump and showing an efficiency of 85 per cent. The bulletin contains many excellent illustrations, capacity and efficiency curves and tables.

Form 3311, Ingersoll-Rand Co., covers Imperial Type X Duplex Steam Driven Compressors suitable for general industrial application of compressed air. The catalog gives tables of sizes and capacities and contains illustrations showing the machine in detail.

Form 8507, Ingersoll-Rand Co., describes the "Little David" Pneumatic Drills. Many types and models are shown, with recommendations as to the particular character of work for which each is adapted. Each tool is illustrated, and tables give sizes and capacities.

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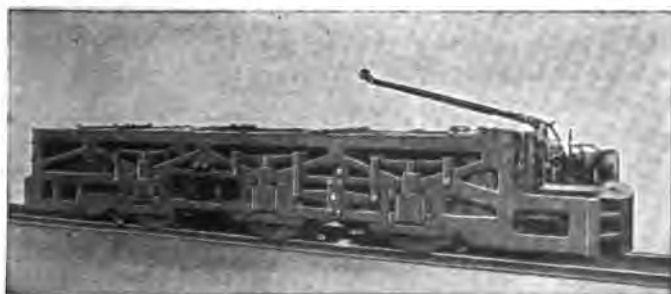
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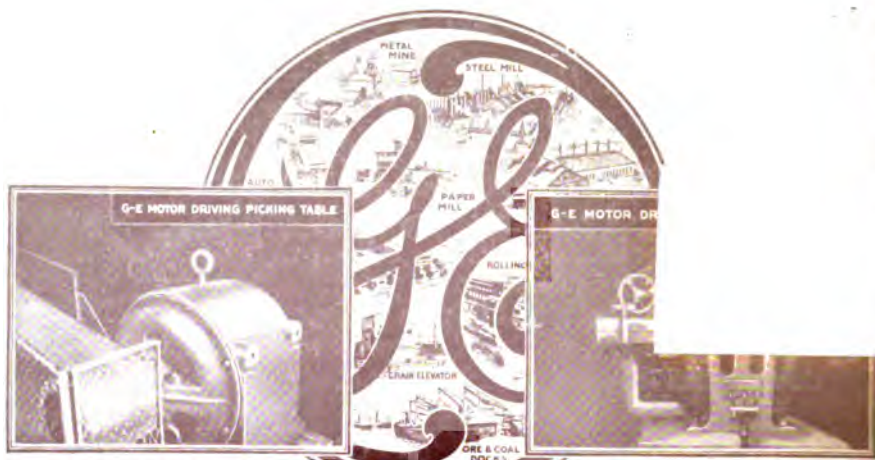
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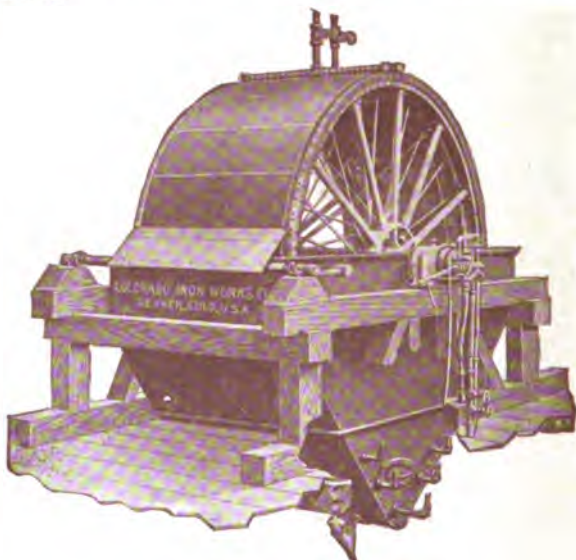
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# AMERICAN INSTITUTE OF MINING ENGINEERS

29 West 39th Street, New York, N. Y.

## PROPOSAL FOR MEMBERSHIP

Mr. \_\_\_\_\_

Address \_\_\_\_\_

is hereby proposed by the undersigned, as a 

|   |               |
|---|---------------|
| { | Member        |
|   | Associate     |
|   | Junior Member |

  
of the American Institute of Mining Engineers.

|       |   |  |
|-------|---|--|
| _____ | { | Signatures of<br>three Members<br>or Associates. |
| _____ |   |  |
| _____ |   |  |

Place of birth \_\_\_\_\_ Year of birth \_\_\_\_\_

Education, when, where and how acquired, with degrees, if any.  
A technical education is not requisite for membership.

| Dates |  |
|-------|--|
|       |  |
|       |  |
|       |  |
|       |  |

Please give record of experience on following page with dates; for example,  
in the form used by "Who's Who In America" and similar publications.

*Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)*

[illegible]

*Present position, and official title, if any:* \_\_\_\_\_

(If the applicant has previously been proposed as a Member, Associate or Junior Member of the Institute this fact should be stated.)

I agree, if elected, to accept election within four months, and to abide by the Constitution and By-Laws

*Signature of Applicant:* \_\_\_\_\_

**Dated** \_\_\_\_\_ **191**

## EXTRACTS FROM THE CONSTITUTION.

## ARTICLE II.—MEMBERS.

**SEC. 1.** The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. \* \* \*

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: **as Members**, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; **as Associates**, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

**As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. \* \* \***

\* Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.



# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

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JULY

1917

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SOFT GROUND MINING, JOPLIN DISTRICT.



PILLARS IN A SHEET GROUND MINE, WEBB CITY DISTRICT. (*Frontispiece.*)

# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

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JULY

1917

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## WESTERN TRIP OF PRESIDENT MOORE

A new departure of unusual importance in Institute annals was inaugurated by the trip of President Philip N. Moore to the Local Sections at Nevada, Southern California, San Francisco, Seattle, Spokane, Montana, Utah, and Colorado. Everywhere the visit of the President was the occasion of an enthusiastic and successful meeting, which is reflected in the accounts published under the different Sections in this and other Bulletins. The President also joined the members of the Columbia Section in their visit to Nelson, British Columbia, at the International Mining Convention, which was attended by mining engineers of Canada and the United States. The President of the Canadian Mining Institute, as well as the officers and members of the Western Section also attended this convention. For full particulars we recommend to our readers the account of this very interesting meeting, which is given on another page of this Bulletin. It is believed that this is the first meeting between the mining engineers of America and any other country since America joined the Allies.

The Secretary of the Institute accompanied the President at the first four meetings and was expecting to attend the others also, but was called home by a death in his family.

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## HOOVER MAKES APPEAL TO STOP FOOD WASTE

We are glad to print the following letter from Herbert C. Hoover, who has recently been appointed Food Administrator, by President Wilson. Mr. Hoover has a tremendous task before him, and we are sure that min-

ing engineers will give him their hearty support by responding to his appeal to stop the waste of food.

*Mr. Hoover's Letter*

In the great National food crisis which confronts us, a crisis so great that the President himself has taken it up, I ask for your personal interest and assistance.

The world is in want of food. The wheat crop of 1916 was short. The crop of 1917 will be shorter still, while the demands of armies and the waste of war require enlarged consumption. Thus with increased need and diminished supply, the world stock of food has reached a point lower than was ever known before in modern times.

From Canada and the United States the Allies are asking 550,000,000 bushels of wheat this year; yet unless special measures are taken the prospect is that both together will be able to give them only 300,000,000. But if wise and sufficient measures are adopted, and then backed up by the patriotic good will of our people, there will be food enough for ourselves and for our Allies as well.

The present excessive cost of food is not justified. There is no corresponding shortage of supply in America. Nevertheless, the situation is very serious. The duty is laid upon us to raise not only food enough for ourselves, but so much more to send abroad that in spite of the submarines enough may reach France and England to keep them supplied. Unless we can feed both our own people and the men who are fighting for us across the water, the great cause which we have joined will be lost.

To meet the needs of the war and of the world, we must produce generously, give freely to our Allies, ourselves eat as much but no more than we need, and especially save the waste. Lasting disgrace will fall upon us if lack of self-restraint should prevent us from taking our proper part in this great conflict against the doctrine of "might makes right."

The waste in food amounts at least to fifty dollars a year for every family in America. The waste of a single ounce of food each day in every home means a yearly waste of nearly 500,000,000 pounds of food. The waste of a single slice of bread each day in every home means the daily waste of a million loaves of bread. The thousand million dollars of needless waste which thus takes place yearly in the households of America can and should be stopped.

The ways of thrifty living are not new. In order to practice them we need do little more than return to the frugal habits of earlier days. To purchase wisely; to lose nothing for lack of proper care; to serve no more than hunger demands; to serve again the unused portion; to preserve in time of plenty against the lean months ahead; to save wheat for France and England by eating more corn ourselves, because they do not bake at home, and corn bread is unknown; these are some of the means of helping to save the waste and win the war. I suggest them in coöperation with the U. S. Department of Agriculture, to which, as well as to the County Agricultural Agents, the Home Demonstration Agents in the cities, and the State Agricultural College, inquiry may profitably be made for printed and written advice about saving food.

Will you join in the good work of saving more food for the women and children of Belgium, the orphans of France, the men in the trenches who are fighting our battles, and for the people of the United States?

The women of America have never failed to answer such a call as comes to them now. The saving of food is within their sphere. Without food conservation we cannot win the war. The woman who conserves the food supply of America and her Allies renders a higher service to her country and to all humanity as well. In very truth, the outcome of the world war is in the hands of the women no less than in the hands of the men.

Again I ask for your coöperation. Let us work together in whole-hearted response to the patriotic duty to which we are called. Let us make it the purpose and the motto of every household in America to *Save the Waste and Win the War*.

Yours faithfully,  
HERBERT HOOVER.

### AMERICAN COMMITTEE OF ENGINEERS IN LONDON

A letter has been received by the Secretary of the Institute from Mr. C. W. Purington, Honorary Secretary of the American Committee of Engineers in London, whose offices are at 6, Copthall Avenue, London, E. C. 2, which will be of interest to any member who may be intending to go to England. Mr. Purington says:

"A Committee of Americans has been formed here, consisting mostly of Mining Engineers, and those interested in mining and allied industries. We should be very glad, among other duties, to give assistance to any American engineers arriving in England on official business connected with the war. This assistance would probably take the form of offering advice and hints as to saving time in approaching British official Departments and generally in getting about."

This Committee of Engineers was organized with the approval of the American Ambassador to Great Britain, who has recommended its services to the United States Government in connection with the conduct of the war.

### PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, MAY 25, 1917

The following Nominating Committee having been already appointed and approved by the Executive Committee in accordance with authority vested in the Executive Committee by the Board of Directors, was approved by the Directors: John V. N. Dorr, Chairman, Percy G. Beckett, J. W. Hutchinson, Julian Kennedy, William H. Linney, Charles W. Merrill, and E. Gybbon Spilsbury.

The question of the purchase of Liberty Bonds, as well as that of bringing to the attention of the members the advisability of the purchase of these bonds, was referred to the Finance Committee.

Sixty Members, four Associates, and thirty-six Junior Members were elected.

Two Members were reinstated.

Eight resignations were accepted.

#### *Minute*

The following minute was then passed regarding the appointment of Herbert C. Hoover as Food Controller of the United States:

This Board has learned with pleasure that the President has nominated Herbert C. Hoover as Food Controller of the United States.

Mr. Hoover has been for 21 years a member (for 2 years a Vice-President), and is at the present time an Honorary Member, of this Institute, and his professional career is well known to American and foreign engineers. His field work on State and Federal geological surveys; his management of important mining and engineering enterprises in the United States and Australasia, and in China as Chief Engineer of the Imperial Bureau of Mines; his courageous and skillful leadership of a continuous battle of 39 days, and his subsequent organization and conduct of the defense of Tientsin, besieged by the Boxer forces; his directorship, on his own responsibility, or as a member of the firm of Bewick, Moreing & Co., of gigantic works of engineering and transportation in almost every part of the world; had marked him, before the beginning of the present war, as one of the most efficient, resourceful and successful administrative engineers of his generation.

These qualities were strikingly exhibited when, at the outbreak of the war, Mr. Hoover organized in London a volunteer committee of American mining engineers, which supplied the immediate needs and returned in safety to this country 100,000 frightened and helpless American refugees. It has been asserted that this great achievement was due to the support of the U. S. Government. But that Committee expended in its work \$300,000 of the money of its members and their friends, and commanded the support of the Government by its conspicuous success.

In like manner, that wonderful achievement, in an emergency which found all governmental agencies inadequate and unprepared, led to the selection of Mr. Hoover as Head of the Belgian Relief Commission, which has expended more than a hundred million dollars with unexampled economy, honesty and efficiency, retaining the confidence of both belligerent parties.

In view of the foregoing record, and of its own knowledge of Mr. Hoover's personal character and methods, this Board unhesitatingly declares its opinion that he is the man best qualified to meet the emergency which has suggested the action of the President, and earnestly recommends to the Congress of the United States the approval of that action by appropriate legislation, and to the Senate of the United States the ratification of Mr. Hoover's appointment as Food Controller.

---

## ST. LOUIS MEETING

Oct. 8-13, 1917

### COMMITTEE ON ARRANGEMENTS

H. A. BUEHLER, *Chairman*,  
C. J. ADAMI, *Southeastern*,  
F. W. DEWOLF, *Book*,  
L. L. HUTCHISON, *Tulsa*,  
P. N. MOORE, *Reception*,  
J. D. ROBERTSON, *Publicity*,  
H. A. WHEELER, *Excursions*,

W. E. McCOURT, *Secretary*,  
J. A. CASELTON, *Entertainment*,  
E. F. GOLTRA, *Finance*,  
E. McAULIFFE, *Transportation*,  
MRS. P. N. MOORE, *Ladies*,  
VICTOR RAKOWSKI, *Joplin*,  
A. THACHER.

## PROGRAM

*Monday, October 8*

Morning: Technical Session.  
Afternoon: Boat ride to Herculaneum, Mo.  
Evening: Dinner on boat and social evening.

*Tuesday, October 9*

Morning: Visits to plants in the vicinity of St. Louis and coal mines in Illinois.  
Afternoon: Technical Session.  
Evening: Banquet.

*Wednesday, October 10*

Trip by special train to the Southeast Missouri lead district.

*Thursday, October 11*

Morning: Visits to plants in the vicinity of St. Louis.  
Afternoon: Technical Session.  
Evening: Leave by special train for Joplin and Tulsa.

*Friday, October 12, and Saturday, October 13*

A choice of one of three possibilities, noted below, will be given:

1. Two days in the Tulsa area, including trips to an oil pool, refineries, gasoline plants, and a technical session.
2. Two days in the Joplin-Miami district, including surface and underground trips, visit to the Pittsburg, Kans., coal field, and a technical session.
3. One day in the Joplin-Miami district and one day in the Tulsa area.

## MINERAL RESOURCES OF THE LOWER MISSISSIPPI VALLEY

This area includes the largest lead district in the world, producing about one-seventh of the total lead. The mines are monstrous caverns, often with roofs 100 feet high. The ore which is found in magnesian limestone, varies in richness, but averages 3 per cent. lead when reduced to metal. The mills are large and show interesting developments in flotation.

In the smelters, the shaft furnace and the modified hearth processes of reduction are used. There are also white lead works of the latest design. It is planned to furnish opportunities to visit all of these by special trip to Southeast Missouri and other trips to smelters.

Equally important is the zinc district of Oklahoma, Missouri, Kansas and Arkansas. This is also the largest of its kind in the world, producing, now, nearly one-third of all the zinc. The mines are in limestone and chert and ore containing as low as 1 per cent. zinc is mined, cleaned and smelted. One of the gas smelters in Kansas has a record of continuous operation for ten years with the exception of 30 days, and is still running. A special trip to this district is planned where the Joplin mines and the new developments at Miami, Oklahoma, will be seen. Some of the smelters will also be visited, in some of which the process of the manufacture of sulphuric acid from the roast fumes may be seen.

The coal mines of Illinois, Missouri, Kansas, Oklahoma and Arkansas furnish fuel over all of the central region of this country. Visitors may see enormous hoisting plants and elaborate underground electric tram systems, with underground repair shops for rewinding armatures. Some of these mines have produced as much as 5,500 tons in eight hours and the total output of Illinois in 1916 was 63,000,000 tons, of which the district tributary to St. Louis produced 44,000,000. Special trips to this district are proposed, as well as to Pittsburg, Kansas, to see the interesting steam shovel system of coal stripping.

Everyone has heard of the enormous oil district developed in Oklahoma, with Tulsa as a center. In 1916, the United States produced nearly 292,000,000 barrels of which 38 per cent. came from the Kansas-Oklahoma field. Visits will be made to gasoline plants and refineries and also to the Cushing and Glenn pools where interesting geologic features will be noted. Considerably less in size, but with a very great interest to St. Louis and vicinity, is the oil and gas district now being developed in Eastern Illinois. The possibilities of bringing natural gas into St. Louis are promising.

Though the steel works in this district cannot compete in size with those in Pennsylvania, nevertheless, St. Louis and East St. Louis are the largest producers of steel castings in the United States, if not in the world. The location of St. Louis, with its numerous railroads and the revived interest in the river traffic, makes it a natural center for steel manufacture. The smelting of iron has been resumed and it is believed that, with water transportation, iron ore and fuel can be brought at reasonable prices.

The excellent character of the fire clays in St. Louis has caused the brick to be known all over the United States and Mexico. Retort clays, glass-pot clays, pure kaolin, and other clays are found in comparative abundance in this district.

Building stones of many varieties are widely distributed, including the red, monolithic granite and the almost marble like limestones. There is also a large production of lime and hydrated lime. Add to these some of the most successful cement plants in the United States and it will be seen that there is no lack of structural material. Gypsum is also mined in large quantities in Kansas.

There is an array of other minerals being exploited or undergoing development. Barytes, for example, is the basis of lithopone, for which there has been a very large demand in the past few years. Missouri is the largest producer of barytes in the United States, having furnished 70,000 tons last year. Nickel and cobalt occur in Missouri and two companies are developing them. Tungsten is another rare metal now being mined. Copper is also produced in Southeastern Missouri. Manganese occurs to some extent in Missouri and several companies in Arkansas are mining it. Some of the purest deposits of salt are being worked in Kansas. Fluorspar is a large product of Southern Illinois and Northern Kentucky and excellent grades of bauxite are mined in Arkansas. Tripoli, a very light porous, decomposed flint rock, is extensively worked in Southwestern Missouri.

A book describing the resources of the region is in preparation and will be ready for distribution at the October Meeting.

J. D. ROBERTSON, *Chairman,*  
Publicity Committee.



## THE WOMAN'S AUXILIARY TO THE A. I. M. E.

MRS. SIDNEY J. JENNINGS, *President*,  
 MRS. ARTHUR S. DWIGHT, *First Vice-President*,  
 MRS. KARL EILERS, *Second Vice-President*,  
 MRS. H. W. HARDINGE, *Third Vice-President*,  
 MRS. BRADLEY STOUGHTON, *Recording Secretary*,  
 MRS. AXEL O. IHLSENG, *Corresponding Secretary*,  
 MRS. ARTHUR S. DWIGHT, *Acting Treasurer*.

*Welfare Committee*

MRS. C. C. BURGER, *Chairman*,

*Sub-Committees**Americanization**Belgian Relief*

MRS. LEVI HOLBROOK, *Chairman*,      MRS. HENRY H. KNOX, *Chairman*,

*Emergency*

MRS. HERMAN A. PROSSER, *Chairman*.

Since the annual meeting held in February, 1917, when the Woman's Auxiliary to the A. I. M. E., was formed, the work of organization, which of necessity must be slow, has steadily gone forward. Under the Chairmanship of Mrs. C. C. Burger of the Welfare Committee, three active sub-committees have been formed: that of Americanization, under Mrs. Levi Holbrook, who is eminently fitted for such a position, by her active and valuable work in a number of the fine patriotic societies of our country; the Belgian Relief Committee, under Mrs. Henry H. Knox, which has done phenomenal work in the three months of its existence, and whose report is later given in detail; and the Emergency Committee under Mrs. Herman A. Prosser, which has a fund already established, and is in close touch with Major Arthur S. Dwight, ready to act when necessity arises.

The officers of the Auxiliary feel more than gratified at the response so far received, and in a later issue will print extracts from letters, if permission can be gotten from the writers, showing the interest and enthusiasm that have been aroused by the formation of a Woman's Auxiliary. Answers have even come from far Korea "the hermit nation," from Australia, from South America, showing what a bond of sympathy and helpfulness is in process of growth.

It is our sad duty to announce the death of our Treasurer, Mrs. George D. Barron, who, from the very inception of our plan to organize, was an inspiring and valuable co-worker. Her place will be difficult to fill.

Mrs. Arthur S. Dwight is Acting Treasurer at present.

The Auxiliary wishes to express its heartiest thanks to Mr. Stoughton, the genial and efficient Secretary of the A. I. M. E., for his coöperation and valuable suggestions in the difficult work of organizing. He has indeed been "a guide, philosopher and friend."

SUSANNE M. IHLSENG (MRS. AXEL O.),  
*Corresponding Secretary.*

*Report of the Sub-Committee on Belgian Relief*

In accordance with a resolution passed at a meeting of the Welfare Committee of The Woman's Auxiliary to the A. I. M. E. on March 8, a Sub-Committee to work for Belgian Relief was formed as follows:

MRS. HERBERT C. HOOVER, *Honorary Chairman*,  
MRS. HENRY H. KNOX, *Chairman*,  
MRS. AXEL O. IHLSENG, *Secretary*,  
MRS. LOUIS D. HUNTOON, *Treasurer*,  
MRS. H. W. HARDINGE,                      MRS. HARRIS K. MASTERS,  
MRS. W. L. HONNOLD,                      MRS. BRADLEY STOUGHTON.

The policy of the Committee was to form as many local Sub-Committees as possible in the mining centers of the United States and South America. In response to a circular letter, ten Sub-Committees have been formed under:

MRS. ARTHUR HOULE, Bisbee, Ariz.  
MRS. SCOTT H. SHERMAN, Christmas, Ariz.  
MRS. R. C. GEMMELL, Salt Lake City, Utah.  
MRS. J. H. McCORMICK, Marysville, Mont.  
MRS. F. P. SINN, Palmerton, Pa.  
MRS. LOUIS S. CATES, Ray, Ariz.  
MRS. E. M. MARSHALL, Globe, Ariz.  
MRS. HENRY G. McCLAIN, Telluride, Colo.  
MRS. L. V. BENDER, Anaconda, Mont.  
MRS. WALTER FITCH, SR., Eureka, Utah.

They have been an inspiration to this Committee by their generous support and efficient work.

A letter was received from the Secretary of the Mining and Metallurgical Society inviting this Committee to act as a Woman's Committee for the Belgian Relief work of the Mining and Metallurgical Society, which the Committee gladly consented to do.

At a meeting of the League for Political Education held at Carnegie Hall on March 26, Mr. Garda, who had just returned from Belgium, in a ten minute address made such a fervent appeal for stricken Belgium that in money and pledges about \$3,000 was subscribed. Through this Committee and the Sub-Committees, about \$12,000 has been raised, which means that twelve thousand Belgian children have been fed for one month with a supplementary meal for which Mr. Hoover asked.

A letter from the Secretary of the Commission for Relief in Belgium announced that the necessary contributions for the Relief in Belgium and Northern France would be taken over by the United States Government, payments to commence on June 1, and also said that they would be glad to have remittances up to that date but made no appeal for contributions thereafter. It asked, however, that the effective organization of the Committees interested in Belgian Relief should be kept alive as far as possible.

In view of the instant need of Belgium having been met by the United States Government, this Committee was authorized by the Welfare Committee to undertake some other branch of Foreign War Relief.

The Committee has withheld its decision. In a multitude of objects all making a strong appeal to our sympathies, it is difficult to determine for which one we will work. As the U. S. Army Engineers are about to undertake the task of reconstruction in Northern France, to aid in that work might seem the best field for our activities.

One of the great needs in that devastated country is to bring the French peasant or laborer and his family back to the homestead. This can only be done by supplying the tools for the artisan and agricultural instruments for the farmer. This would be working on constructive lines of helping people to help themselves, which is the earnest desire of the French Government, and would be following the general line upon which the engineers are working.

This Committee extends its hearty thanks and appreciation to the women who have so splendidly responded to its appeal and trusts that in its new work it will have the same generous support.

SOPHIE C. KNOX, (MRS. HENRY H.)  
*Chairman.*

**MEMBERS OF THE INSTITUTE IN MILITARY SERVICE**

BARBOUR, PERCY E., Deputy Supt. (Captain), New York State Troopers; Captain, 22nd Regiment, N. Y. N. G.

BARLING, H. B., 1st Lieutenant, Engineer Officers' Reserve Corps.

CHAPMAN, R. H., Major, Engineer Officers' Reserve Corps.

DUNN, THEODORE S., 1st Lieutenant, Engineer Officers' Reserve Corps.

DWIGHT, ARTHUR S., Major, Engineer Officers' Reserve Corps.

FORBES, CARROLL R., Captain, Engineer Officers' Reserve Corps.

GOODSPEED, G. E., JR., 1st Lieutenant, Engineer Officers' Reserve Corps.

GRAVATT, C. MARSHALL, Aviation Corps, Columbus, Ohio.

GRUNOW, W. R., 1st Lieutenant, Engineer Officers' Reserve Corps.

GRUNSKY, CARL E., JR., Captain, Engineer Officers' Reserve Corps.

HAGUE, WILLIAM, 1st Lieutenant, Engineer Officers' Reserve Corps.

HEGDEM, ALFRED G., Captain, Engineer Officers' Reserve Corps.

HINES, P. R., 1st Lieutenant, Engineer Officers' Reserve Corps.

HURD, RUKARD, Major, Engineer Officers' Reserve Corps.

IRVING, J. D., Captain, Engineer Officers' Reserve Corps.

KNAPP, ARTHUR, Captain, Engineer Officers' Reserve Corps.]

LACROIX, M. F., Captain, Engineer Officers' Reserve Corps.

LEONARD, W. H., Captain, Troop E, 1st Regiment, Colorado Cavalry.

MEWHIRTER, SYDNEY A., Designate Lieut., Corps of Engineers, U. S. Reserve.

MUDD, SEELEY W., Major, Engineer Officers' Reserve Corps.

PARKS, HENRY M., Captain, Engineer Officers' Reserve Corps.

ROUDABUSH, M. A., 2nd Lieutenant, Engineer Officers' Reserve Corps.

SACKET, CHARLES T., Captain, Engineer Officers' Reserve Corps.

STEHLI, H. J., Captain, Engineer Officers' Reserve Corps.

VALLAT, BENJAMIN W., Major, Battalion of Engineers, Washington College of Mines.

VAN NESS, W. WATERS, Captain, Royal Engineers, British Army.

WEINBERG, GEORGE S., Major, Engineer Officers' Reserve Corps.

WORTH, JOHN G., Captain Quartermaster, Officers' Reserve Corps.

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**Request for Information.**—This list is no doubt incomplete, and we would appreciate information concerning members who are engaged in any military service.

## PRESENTATION OF THE JOHN FRITZ MEDAL TO HENRY MARION HOWE

The John Fritz Medal is awarded from time to time for notable scientific or industrial achievement by a board composed of representatives from the four engineering Societies.

The members of the Board of Award for 1917 are:

### A. S. C. E.

George F. Swain,  
Charles Warren Hunt,  
Charles D. Marx,  
Clemens Herschel.

### A. S. M. E.

Ambrose Swasey,  
John A. Brashear,  
Frederick R. Hutton,  
John R. Freeman.

### A. I. M. E.

E. Gybbon Spilsbury,  
Charles F. Rand,  
Christopher R. Corning,  
Benjamin B. Thayer.

### A. I. E. E.

C. O. Mailloux,  
Paul M. Lincoln,  
John J. Carty,  
Harold W. Buck.

The John Fritz Medal was awarded in January, 1917, to Dr. Henry Marion Howe, for his "investigations in metallurgy, especially in the metallography of Iron and Steel."

The presentation ceremonies were held in the Auditorium of the United Engineering Societies' Building, New York, on Thursday evening, May 10.

Mr. Ambrose Swasey, Chairman of the John Fritz Medal Board of Award, presided, and addresses were delivered by Dr. Rossiter W. Raymond, Secretary Emeritus of the American Institute of Mining Engineers, and Dr. Ira N. Hollis, President of the American Society of Mechanical Engineers.

Professor Albert Sauveur, Chairman of the Board at the time the award was made, presented the medal, and the ceremonies concluded with the response of Dr. Howe.

### *Address of the Chairman*

Members of the Engineering Societies and Friends: It is indeed a happy privilege to meet here this evening to honor one who has gained distinction as a scientist and engineer. In honoring him we are also paying tribute to the memory of that noble life and character for whom the medal to be presented was established.

The awarding of the John Fritz medal has had an elevating and broadening influence; it has increased the spirit of cooperation between the engineering societies, and because of it the engineering profession of our country occupies a higher plane and has a broader horizon than ever before.

I am sure no one could take a greater pride in what has been accomplished than Mr. Fritz, for he was, as we know, deeply interested and was most helpful in every way in working out the plans for the establishment of the medal.

The recipients of the medal form a most notable honor roll. Many of them have finished their work and have passed on, but their achievements and the influence of their lives still live. Others are continuing their work, and we hope they may be spared for years to come. It is especially gratifying that we have with us this evening Mr. Robert W. Hunt, one of the earlier recipients of the medal. As it is always an inspiration to hear their names, I will read them: John Fritz, Lord Kelvin, George Westinghouse, Alexander Graham Bell, Thomas Alva Edison, Charles Talbot Porter, Alfred Noble, Sir William Henry White, Robert Wilson Hunt, John Edson Sweet, James Douglas, Elihu Thomson, and this evening to be added, Henry Marion Howe.

Taking up the ceremonies of the presentation, I will first call upon one for whom we have a deep affection. Because of his many years of splendid work for the engineering profession, you all know him, and he needs no introduction. I am pleased to present to you Dr. Rossiter W. Raymond.

#### *Address of Dr. Raymond*

The genesis of a new science or branch of science is on this wise:

First, there is the invention of some new means or method of investigation, such as the chemical laboratory, the telescope, the microscope, or the spectroscope, followed at once by an innumerable host of startling, confusing, mostly qualitative observations. Such an invention reveals to mankind a dazzling but vague new world, the very existence of which was scarcely suspected before, and the indefinite outlines of which inspire poetic generalizations and vast hypotheses; beautiful, if true—beautiful, indeed, irrespective of their truth.

Then comes the long period, less thrilling yet not less meritorious and even more important, during which the means and methods of observation are rendered more precise, and the recognition of differences more minute. At the same time, the older data are revised, corrected and sorted; many, probably most, of them being rejected. This is the stage of preparation of the raw material of the science; and it is characterized in every science by the accumulation of a well-nigh intolerable burden of distinctly separated facts. But this burden must be patiently borne, and the separate units of which it is composed must be delimited, measured and weighed, in order that the mathematical, that is, the quantitative, character may be impressed upon them.

Illustrations of this principle will occur to all who have followed the growth of any science.

Gen. Myer, the creator of our U. S. weather bureau, once told me that when he began to organize that service at Washington, he found in the garret of the Smithsonian Institution tons of old papers, containing "weather observations," contributed by public-spirited citizens all over the country, who had heroically got out of bed at certain hours to read their thermometers, rain-gages and weather-cocks and anemometers. But nobody had tested or standardized their apparatus, and nobody

guaranteed its freedom from accidental aberration or mischievous meddling.

I have heard also of a great German mineralogist, who made many thousand measurements of crystals, and based upon their varying angles some beautiful hypotheses. But after his death, another mineralogist tested his goniometer, and found in it defects which vitiated his observations.

Such are the pseudo-materials of science, which must be got out of the way before safe induction can begin. But even really scientific data



HENRY MARION HOWE.

may present, during the stage of verification and inchoate arrangement, an overwhelming complexity which threatens to smother a new science in its cradle. Take, for instance, the condition of biology before Wallace and Darwin. Everybody was furiously busy detecting differences, and founding upon them the definitions of new species; and the mere catalogs of species were beyond the student's comprehensive grasp. We all remember the universal sigh of relief with which the scientific world, welcoming the Darwinian hypothesis as its new guide, began to study

similarities and relationships, instead of differences, and saw the incalculable multitude of individual species group themselves in a grand order, illuminated by a new law.

This emergence of a general formula from a chaos of equations marks the final stage in the genesis of a science. That is not the end, but only the beginning, of its history. It must still grow by accretion, subdivide by fissile separation, and establish its relations to other sciences. And at every step the operations of observation, verification, criticism, analysis and synthesis must be repeated, just as the embryo rehearses in miniature the history of a species.

It follows that the stages of development which I have indicated, though they may seem, on the large scale, to follow one another, really go on, *pari passu*. There is no time at which any one of them ceases. Yet with regard to a single and limited branch of scientific inquiry, their existence and succession may be clearly discerned.

It is my difficult but honorable and welcome task to set before you an outline of the work of a master who has contributed mightily to all these phases in the growth of the new science of metallography: as a discoverer and observer; as an industrious compiler; and as the builder of a noble edifice out of the materials thus gathered and prepared.

It is not my purpose to give a complete biographical and bibliographical statement. I shall attach to the published text of these remarks an appendix containing in condensed form the dates and nature of Prof. Howe's technical positions, the titles of his principal publications, and the degrees and distinctions conferred upon him. This will supply the deficiencies of the sketch which I hastily prepared for the John Fritz Medal Book.

But I must say here at least so much as will indicate the peculiarly fitting endowment and environment by which this life and its work were determined and perfected.

Henry Marion Howe was born March 2, 1848, at Boston, Mass. His father was Dr. Samuel G. Howe, famous for his service to Greece in her war for independence (from 1824 to 1830) and later for his labors in the instruction of the blind. His mother was Julia Ward Howe, author of the Battle Hymn of the Republic, and leader in many reforms. It was a good stock on both sides, making him heir to intellectual keenness and refinement, the capacity for both enthusiasm and perseverance, a passion for the pursuit of knowledge, and a gift of clear and felicitous statement.

This inheritance was improved by a liberal education. No matter how vehemently the business or scientific value of a "college course" may be controverted, I notice that, without exception, the successful men who have had such a course are glad they had it, and those who have not, wish they had. We may be sure that Prof. Howe's easy command of his field in technical literature owes much to the circumstance that he was graduated in 1865 from the famous Boston Latin School, and, four years later, received his degree as Bachelor of Arts from Harvard College. Thus equipped, he entered the Massachusetts Institute of Technology, which gave him in 1871 the degree of "Graduate in the Department of Geology and Mining Engineering"—a cumbrous title for which the institution substituted, a few years later, that of "Bachelor of Science." And Harvard made him Master of Arts in 1872, and Doctor of Laws in 1905.



But upon this basis of wide and liberal culture, it was necessary to his future achievements that he should lay another foundation of acquaintance with practice; and this he did during the next dozen years, as a student in the steel works at Troy, manager of works at Pittsburgh, and designer and builder of the works of the Orford Nickel and Copper Co. at Capelton and Eustis in the province of Quebec, and at Bergen Point, New Jersey. His experience in the metallurgy of copper bore legitimate fruit in the publication at a later period of his first book, *Copper-Smelling*; but before this appeared (in 1885), he had already become known through his technical papers as an acute observer and reasoner, by no means averse to friendly controversy.

Since it is my present duty to offer, not so much a symmetrical and complete account of Prof. Howe's activities as a description of the particular labors in recognition of which he receives today from his brother engineers the John Fritz gold medal, I shall pass lightly over the general features of his career, merely observing that from 1883 to 1897 he resided at Boston, and, besides his private practice as a consulting metallurgist and expert witness in metallurgical patent suits, was lecturer on metallurgy at the Massachusetts Institute of Technology; that in 1897 he was called to the professorship of metallurgy in Columbia University, from which position he retired in 1913 with the title of Professor Emeritus; and that for the last ten years he has declined as far as practicable all professional business, in order to devote himself exclusively to the completion of what had clearly become the great scientific mission of his life. To this end, he has maintained at his own expense a special laboratory of research.

It gives me pleasure to fancy, whether or no the fancy be also fact, that I remember the beginning of that mission, forty-odd years ago. In 1871, the first year of the existence of the American Institute of Mining Engineers, Mr. Howe, then just graduated from the Massachusetts Institute of Technology, became a member of the new organization. His two earliest contributions to its *Transactions*, Blast-Furnace Economy, in Vol. III, and Thoughts on the Thermic Curves of Blast-Furnaces, in Vol. V, indicated already that he was making a scientific study of practice. But between the dates of these papers a famous controversy was inaugurated in the forum of the Institute by the brilliant paper of Alexander L. Holley, *primus inter pares*, entitled "What is Steel?" In this controversy, eminent metallurgists like Wedding, chemists like Prime, and expert captains of industry like Metcalf, vigorously took part; but of all the knights of the tourney, none rode a straighter course or laid in rest a sharper lance than Howe, whose paper on The Nomenclature of Iron ardently advocated a scientific as distinguished from a commercial nomenclature. To tell the truth, the battle had a commercial origin. The real question at issue was not, What is steel? but, What may fairly be called steel at the custom-house? And the technical reputation of the Institute was invoked by one party in favor of a convenient, practical industrial classification which would relieve importers from the expensive refinements of science.

Well, there was even an International Committee on the subject, and the Committee made a Report, and the Report was not adopted by the Institute, except provisionally for optional use in its publications, because the Institute was wisely deemed to be an arena of discussions and not a tribunal of decisions. Never was a more fortunate position

assumed, for the great question was left open, as it should be always. The tariff difficulty was adjusted somehow; the antagonists shook hands; the thunder of the captains of industry and the shoutings died away. But one man continued the inquiry fiercely in his own soul. Henry Marion Howe, before whom the dust of the controversy loomed like a giant Afrite still defying attack, devoted his life thenceforward to the mighty conundrum, What is Steel? That he has spoken the last word—even his own last word—on the subject, no one would venture to say; but certain it is that he has brought us to a degree of knowledge the very existence of which was scarcely dreamed in 1876.

It is curious that at that time he insisted upon the capacity of steel for hardening as an essential element in classification. For it was through this door—the study of the conditions and nature of the hardening process—that the advance was to be made into the wider field of knowledge. And the instrument of knowledge was to be the new science of metallógraphy.

That science, at least so far as it relates to iron and steel, may be said to have begun with the observations of Sorby on the microstructure of iron, reported in 1864 and 1868. Martens published independently in 1878. But already in 1868, Tschernoff had enunciated the chief laws that govern the metallography of iron. These were supplemented by the appearance in *Stahl und Eisen*, 1885, of Brinell's laws. All these creditable steps of progress were rendered more or less uncertain and incomplete by the imperfection of the apparatus and methods of precision, by consequent errors of observation, and by gaps in the data—though the aggregate quantity of material was already overwhelming. Then came, in 1887 and subsequent years, the remarkable investigations and intuitions of Floris Osmond, who discovered that metals frequently combine to form definite chemical compounds, and that these compounds frequently form solid solutions. He discovered also the thermal critical points of iron, and, interpreting these changes in the cooling-curve as indications of some molecular change, propounded the brilliant hypothesis of the allotropy of iron which furnished the acceptable allotropic theory of the hardening of steel. To him, as Prof. Sauveur remarks in his Biographical Notice of Osmond (*Trans.* (1913), 45, 274), we owe likewise the discovery of austenite, the non-magnetic solid solution of iron and carbon existing above the thermal critical range, and the transition-constituents, martensite, troostite and sorbite, marking as many distinct and important steps in the transformation, on cooling, of the solid solution austenite into the ferrite-cementite aggregate.

This brings us to the date of Howe's first book on the subject; and the history of the new science, as I have already sketched it in general terms, is epitomized in the essays and books of Prof. Howe from the appearance of his *Metallurgy of Steel* in 1888 to that of his *Metallography of Steel and Cast Iron* in 1916. The first of these books was an amazing accumulation of reported facts, tabulated, verified and explained as far as was then practicable. The last is an equally amazing array of facts, but now sifted, tested, logically arranged and luminously interpreted, exhibiting not uncomprehended differences, but significant similarities and relationships. The first was a heap, parts of which had been sorted; the last is an edifice. To produce the first required intelligent and inexhaustible industry and critical discernment. The second exhibits the creative genius of an architect. Between the two lies the history of a

science, to every stage of which this builder has made some important contribution. Let me mention a few of these, under the heads of invention of improved methods; discovery of new facts; testing of data; and correlation and interpretation of observed phenomena. I shall not pretend to comprehensive completeness in this survey. I must be content with the exhibition of typical samples.

One of the new methods invented by Prof. Howe is that of determining the microstructural and other conditions which exist in steel at high temperatures by fixing those conditions through the process of quenching—a method which has been generally adopted, and by the use of which much clearer and more trustworthy results have been obtained than were formerly possible.

Among his discoveries of new facts may be mentioned the isotropy of the effects of plastic deformation (announced in 1888, and supported in 1914); the possibility of effacing blowholes in soft steel by welding (1909); the relations of graphite flakes in cast-iron, representing, not complete discontinuities of the metal, but only the filling by graphite of the interstices in ferrite skeletons; and the crystallography of the slip-planes (*Metallography of Steel and Cast-iron*, 327).

But this investigator kept himself acquainted with the discoveries of others. In 1893, he confirmed by rigorous experimental proof the brilliant hypothesis of Osmond that the transformations in steel represent the alternate acquisition and loss of hardening capacity.

Under the head of testing and verifying data, I would name the rigorous analysis of the so-called Clapp-Griffiths process; the demonstration that the chief benefit of rolling and forging is due to the low finishing-temperature rather than to the degree of mechanical reduction, as formerly believed; and the striking demonstration that the results of one of the most important investigations of the gas evolved from iron were wholly vitiated by the leakage of the apparatus. All these may be found in the *Metallurgy of Steel*, which contains also analyses of the conditions of dephosphorization; of the chemistry of the crucible process, and of the technique of Bessemer practice.

In a remarkable paper on the structure of steel, contributed to the proceedings of the American Society for Testing Materials for 1911, Prof. Howe collected into the form of propositions or laws the observed relations of heat-treatment to the microstructure and mechanical properties of steel. In that paper, he enunciated twenty-three of these laws; and had he then ceased from his labors, his work would have followed him as an imperishable record indelibly carved upon the history of science. But he was destined to go still further during the five years which followed. His latest book, *The Metallography of Steel and Cast-iron*, published in 1916, occupies intellectually a higher plane than its predecessors, by which I mean that it commands a wider outlook, permitting grander generalizations, and the recognition, in due perspective, of causes, effects and relations. This book, in short, exhibits that simplification which I have described as following the stage of maximum complexity in the development of a science. Its very title, coupling steel with cast-iron, expresses the maturer view which includes both in one continuous series, subject to the same laws—a significant conclusion of the controversy which began with the contention that steel was not cast-iron. Not that the difference has been disproved, but that a higher unity has

been shown to embrace it. Such is indeed the normal end of controversy—"peace without victory" through agreement on a higher level.

This book is not a text-book, repeating what is generally accepted already. Covering the field in which others have meritoriously labored, it propounds, suggests and prophesies new truth and new views of truth. It is a product of the two kinds of genius, the genius which consists in an infinite capacity for taking pains, and the genius which with happy intuition surveys, divines, coördinates and interprets the cosmos in the chaos.

That my estimate of the work of our great American colleague, as summed up and set forth in this book, is not merely an expression of personal affection or patriotic pride, let me prove by one or two opinions from eminent authorities.

I would quote first a few paragraphs translated from a review of the book by Prof. Le Chatelier in the *Revue de Metallurgie* (vol. 13, No. 2, March-April, 1916):

"The new work of Mr. Howe is entirely original. It has nothing in common with the numerous treatises on metallography which have appeared during the last ten years. Whoever has read one of them has read them all. Here we have to do with the personal observations and views of the author. As he indicates in his preface, he has written this treatise, not to state the solidly established principles of a young science, but to cause the creators of that science themselves to think; to open before them new horizons. Hence he has not feared to announce theories sometimes a little hazardous. \* \* \*

"This study will make an epoch in the history of the progress of science. It represents many years of uninterrupted research, but for a much longer period it will certainly be consulted by investigators with fruitful results."

The following remarks are taken from a similar review by H. C. H. Carpenter, Professor of Metallurgy at the Royal School of Mines, South Kensington, England.

"It is quite safe to say that Prof. Howe's book will at once take its place as an authoritative, and indeed, classical, exposition of the field of knowledge with which it deals. From whatever standpoint it is judged, it is a great book—great in its power, lucidity, balance, comprehensiveness, and pre-eminently scientific character."

In a private letter, written with reference to the announcement that the John Fritz medal was to be conferred upon Prof. Howe, the same distinguished author says that he believes no award could have given greater pleasure to the metallurgists of Great Britain, who look upon Prof. Howe, not merely as the *doyen* of metallographists in America but as their most distinguished representative; "a worker whose single-mindedness in the pursuit of truth is an example to us all, and whose conscientiousness gives to his publications a character of their own."

Finally, I take the liberty of quoting from a private letter received by me from a no less eminent American, Prof. Albert Sauveur himself, who, as the President of the Board of Award, bestows this medal to-night:

"Prof. Howe, in his recent masterly book on the metallography of steel and cast-iron, reviews exhaustively and examines critically, as he alone can do, every view at all worthy of recognition, dealing with the subject he covers. I believe that any responsible author who has ever expressed a reasonable opinion on any subject dealing with the metal-

lography of iron and steel will find his views recorded and discussed in this book. Prof. Howe then proceeds to weigh with great fairness and extraordinary intelligence and lucidity the arguments or evidence supporting the different views, and draws his conclusions accordingly. His method is that of a mathematician solving a problem in which each factor is given its proper value. In this way, Howe has rendered to metallurgists an inestimable service; and he alone could render it. Where most of us could see only chaos and obscurity, he is able to bring order, and to discover the light that, under his skillful manipulation, soon illuminates the darkest corners. \* \* \*

"This, I think, is his greatest achievement—the marvellous coördination which he has brought into the science of metallography through his genius and his labor, tremendous in quantity, marvellous in lucidity. To him more than to any one else we owe the quick rejection of weak or ill-supported theories, in favor of the survival of the fittest—a process so necessary to the advance of any branch of human knowledge."

To these expressions of individual and representative opinion should be added the numerous formal honors bestowed upon Prof. Howe by institutions and governments in many lands. As the Appendix shows, England, France and Russia have each conferred three of these recognitions, while others have come from Germany and Sweden. And, last but by no means least to his patriotic heart, come two announcements, received since I began the preparation of this address: the one, of his election as a member of our National Academy of Science; the other, of his selection as one of the charter members enumerated in the proposed Act of Congress incorporating the American Academy of Engineers.

Thus the applause of two continents attends the act of the John Fritz Medal Committee in selecting Henry Marion Howe as a recipient of that great honor. And the act has also another significance. It is a recognition on the part of engineers that the microscope and the test-tube have become the tools of engineering. Indeed, when we remember that the strength of materials is the alphabet of every branch of engineering science represented by the givers of this medal, we must admit that the man who with microscope and test-tube analyzes the causes and determines the conditions of that strength is among the greatest of engineers.

While this testimonial is bestowed by engineers, I cannot forbear to utter my satisfaction that it is received by one who is more even than an engineer; whose many-sided patriotic and literary activity is typical of this age, in which engineers are, as St. Paul said of the Gentiles, "no more strangers and foreigners, but fellow-citizens with the saints." If we may say of him, as Johnson said of Goldsmith, "*non tetigit quod non adornavit*," we may almost add, "*non est, quod non tetigit*."

And so we crown to-day with grateful pride, not only the leader, but also the comrade and friend!

### *The Chairman*

The next speaker is one of the busy men of these days, for he has many duties in addition to his work as President of the Worcester Polytechnic Institute and President of The American Society of Mechanical Engineers, but we know that whatever he undertakes he does well. I have much pleasure in introducing to you Dr. Ira N. Hollis.

*Address of Dr. Ira N. Hollis*

It is not only a privilege, but also an honor, to have a share in the award of the John Fritz Medal to Henry Marion Howe for his contributions to applied science. This medal as a recognition of achievement in applied science has completely justified itself if it has served no other purpose than to record in this admirable way the practical service to science. But it does more in calling attention to the great value of research. The popular conception of invention as a mere happy guess or thought has led many into a totally false conception of the method by which applied science has grown. As a matter of fact, practically all of our advancements have been the results of patient study and research, the work of developing a thought always being the harder and better part of it. In calling attention to the method by which Mr. Howe's great work has been accomplished, this meeting serves to educate and to clear the popular mind on the true value of scientific research and of the abstract study of all science. Not one of the recipients of the John Fritz Medal has earned his right to distinction by mere thought. Each one of them has worked, and worked steadily through his life, with some definite end in view, usually without much thought of the money side of his discovery. In this respect Mr. Howe is a worthy successor of all those who have preceded him on this honorable list. His work has been done quietly and modestly without the help from daily papers that has become so disturbing and, in some cases, so disastrous a part of the scientist's achievement in these later years. He is not only a fitting successor to all the men who have received the same honor, but he is also beyond all others of the same family as John Fritz. No professional man has ever earned lasting fame who has not obtained it through the high respect of those in his own profession. It is not the printing press and the public who make the reputation of a great man, but rather his companions in the profession who understand his work. Mr. Howe is thus singularly happy in the friends who have gathered around him tonight to rejoice with him. They will remember and understand well what he has done for the science of metals and for his country as well. As a teacher, a writer, and a scientist, his life is a speaking example of science wisely and unselfishly used in man's service. He has given and published freely his discoveries and his ideas on steel so that all his fellow citizens might benefit alike. There has been no touch of the baneful search for power through money. Without being an announced leader in scientific management, his work in teaching the industries the value of special steels, in adapting the means to the end, has been a great contribution to safety and efficiency.

He has pointed out the path. Sometimes by simply indicating where the path enters the unknown a teacher opens the way and leads others from traditional methods to new and better things. Life is made up of wandering pathways, and some great pathfinder stands at every parting. James Watt was at the beginning of modern industry, George Corliss at the first great departure in the economical use of steam, John Fritz lived to see his work culminate in the great Bethlehem company. All the others on the list of medallists have found themselves at the beginning of an era in their subjects and now Henry Howe joins these immortals who have stood as the great teachers and pathfinders of mankind.

Nothing that I can say here will add to his reputation. That is

written in his books. This great meeting should, however, lead us to a new dedication of ourselves to the kind of service so much needed toward the rational life of man of this planet.

Science has three aspects: first, to discover truth and follow it; second, to employ the forces and materials of nature for the good of all mankind; third, to set us into closer accord with the will of God.

It is not necessary to dwell on these, except perhaps the last, which is the only pure science that we have. In these days of struggle for power, in these days when the white races are using their scientific knowledge to weaken one another, to destroy one another, it is well for us to search our hearts for those things in which science has been found wanting. At times we have made a god of it, with the thought that it alone has brought the human race out of darkness, that it has dispelled the shadow of superstition, and by promoting the well-being of man, it has formed a true civilization. We sometimes forget that no civilization can continue to exist in an unbalanced condition. Science alone would make of this a hard cold world, just as all intellectuality without heart tends to mere machinery. There are always two sides to a shield. Well-being and comfort without the spirit of self-sacrifice destroy rather than promote an enlightened civilization. It is well, therefore, on an occasion like this to give some thought to the religious side of life, and by religious I mean that kindly spirit brought into the world by Jesus Christ 1900 years ago. Nothing that the world has done or that science has produced is comparable with His one precept as the chief motive by which men can live. We may consider this great struggle going on in Europe, into which we ourselves are plunging, as the greatest thing that ever happened in this world, and yet it is insignificant by the side of that appealing life that ended on the cross so many centuries ago. "Love thy neighbor as thyself" is the best inheritance of the race, as it is the chief hope of the future. Science is valuable only in so far as it vitalizes and strengthens that one principle, because we care nothing for new inventions, we care nothing for the comfort of men, and we care still less for the possible support of a greater population, if the soul be dead. Only so far as science promotes true democracy (not political democracy) and that love of men which will make wars impossible and eventually turn the lawyers into farmers is it worth while. In 1900 years it has done something toward making possible Christ's reign on earth. The famine areas of the earth have been reduced by the development of transportation and the communication by telegraph. Where there is a shortage of crops, we can quickly rush aid and supplies, and the world is generally willing to give a small percentage of its gain. It has remained for science to develop the most effective method of destroying the whole yield of a harvest, and plunge it into the bottom of the sea, so that it is only just to say that the human race has developed no international conscience. The organization for relief is still in its infancy, and there is only enough Christianity left to provide help more or less spasmodically. Another direction in which science has promoted Christianity is in preventive medicine, and in hospitals that have been established for all kinds of infectious and inherited diseases.

But, after all, we have just touched the fringes of a real civilization, in which righteousness will be the guiding principle for nations, as well as for men. When the best minds are devoting themselves to the invention and construction of machinery and chemicals for killing men on a great scale,

we are very far from loving our neighbors as ourselves. Science has therefore done as much to enslave as to emancipate, and we have not yet learned that its votaries are worshipping only half of the truth. Unless the engineer and those who profit through his work can be brought to realize that everything produced through science is held in trust for humanity, we shall make no progress. Our country will be plunged more and more into that lethargic, luxurious state which can be cured only by the shedding of blood. Already we feel in this war a certain impulse of regeneration, and we need it. Our country has been called "dollar mad" by Europe. Almost everything is judged from the one single basis of money. A book is good only if it sells. Fortunately there is one place into which this surrender has not penetrated; that is the teaching profession, and Mr. Howe's life well illustrates the kind of altruism that will bring us a happy regeneration, and a more lasting peace than we shall ever get through the blood of war. A better democracy must come through science. Men must learn to be ashamed to accumulate wealth through invention. No privilege by birth and no inheritance through the patent office should stain our democracy. No king can love his neighbor as himself, no great land owner who has the peasant on his farm can ever feel that the creature who toils is exactly like himself. This great war then is a war against false doctrine. Its end must be that greater democracy which teaches men unselfish service to one another, without which the race is not worthy to remain, and without which it ought to disappear. Only so far as science strengthens righteousness will it contribute to the great task before us as a free people.

We see some signs in our country of a true enlightenment. The wealthy desire to give. Our privately owned colleges and schools are supported in large part by generous men who believe that their wealth is a public trust. It is only a gleam of light, however, for the percentage of giving is extremely small, and our problem will not be solved until we attain that enlightenment which makes it a duty for every man to give. We must think of invention and science and wealth as the mere instruments through which the individual is enabled to benefit the race. It is important that we should consider this whole question now, because the treaty of peace that must eventually come will ask for an answer. Science and Christianity must join hands to find some means by which the problem of a larger life can be solved between nations, as between individuals.

Our profession has an ethical purpose, and it will fail if it does not see clearly that its place is not with science alone. Under various names it has been the one great constructive profession since reason began to develop in the creatures on this earth. Its history is practically the history of the human race. The political struggles have but recorded the advance of science in one form or another. To us as a profession, therefore, is confided the great task of enabling men to live on this planet under the theory that the highest happiness is attained through love, rather than through force. Organization is a science, as important as the machine. Life itself is the greatest of sciences. We must contribute our share.

My chief reason for dwelling on the moral aspect of science, especially on that side of it which promotes the practical application, is that Mr. Howe's career well illustrates the motive that must always lie behind the engineer's work, if our profession is to live and to be found worthy.



*The Chairman*

The following message was received by Mr. E. G. Spilsbury, Chairman of Arrangements:

"It is with sincere regret I now find I will be unable to be present on the occasion of the presentation of the John Fritz medal to Professor Henry Marion Howe. I congratulate the Board of Award on its wise selection, and Professor Howe for his splendid achievements.

(Signed) E. H. GARY."

These are strenuous times for our great captains of industry, and we very much regret that Judge Gary could not be with us this evening, for I am sure his message would have been most inspiring to all.

It is especially fitting that the Chairman of the Board of Awards should on this occasion formally present the John Fritz medal to the recipient, and I will therefore now ask Professor Sauveur to formally present the John Fritz gold medal to Dr. Howe.

*Address of Prof. Sauveur.*

It seems to me eminently fitting that this high token of esteem should be presented on the day when the sons of France are being received in this city with so much enthusiasm and affection—when sleeping America shows signs of awakening—for Prof. Howe has been from the start a vigorous and whole souled champion of France and her Allies—his vision has never been clouded—his mind was made up in August, 1914, when the enemies of mankind started the crucifixion of Belgium, which was to be followed by the crucifixion of the rest of the world.

He never believed in a long period of neutrality, he did not believe that all fighting nations were "War mad"—that their aims were the same—that their quarrel was no concern of ours—he never wished for a peace without victory—and I am certain that he believes now that material assistance should be sent at once to bleeding France—that we should not wait until the last drop of new life blood has been spent.

In these tragic days when the fate of the human race is being decided, the services Prof. Howe rendered by his fearless and chivalrous stand loom up before me as big as his services to Science—it is my excuse for alluding to them.

After what you have heard there can be no doubt left—if any doubt ever existed—in anyone's mind as to the wisdom and discrimination exercised by the John Fritz Medal Board of Award in selecting its medalist for 1917.

There is nothing that I could add to what has been so ably—yea, so brilliantly—told by Dr. Raymond. To me, however, as Chairman of this Board at the time the award was made, is assigned the delightful duty of actually presenting the John Fritz Medal to Professor Howe.

Henry Marion Howe, son of an illustrious father and of an illustrious mother,

Lover of justice and of humanity,  
Public servant and public benefactor,  
Master of the English language,  
Loyal and devoted friend,

Untiring and unselfish worker in an important field of science,  
 Stimulating teacher, inspiring investigator and generous collaborator,  
 hundreds have been guided by the light radiating from your pen, encouraged by your kind words, dazzled it is true, but also stirred to further and more vigorous efforts by the heights you reach,

Voyager in realms but dimly perceived by your fellow-workers,  
 Lone explorer of fields destined to yield rich harvest to future generations,

Man of genius, honored and loved the world over,  
 By authority of the Board of Award, I now place in your hands the John Fritz Medal.

*Reply of Prof. Howe*

It is not to me but to the bright unbodied image which you have created, to the foster child of your chivalry and your leniency, that these more than generous words apply, that this noble memorial is due. I thank you from the bottom of my heart on his behalf, while on my own I thank you as warmly for his acquaintance, for the hope that it may kindle into an inspiring friendship, letting me "grapple him to my soul with hoops of steel," and above all for this incitement to further and vigorous striving, holding with Ulysses that

"Old age hath yet his honor and his toil;  
 ————— something ere the end,  
 "Some work of noble note may yet be done."

To prolong thanks is so thankless that I might well now hold my peace were it not for the world crisis, ever in our consciousness. This so presses for our best thought that attention to other matters suggests fiddling while Rome burns. For Rome truly is in flames. We are as a family in a burning home. Democracy itself is at stake.

This conflagration should arouse Americans all the more because it is a necessary and integral part of the train of events which our forefathers started when those

"embattled farmers stood,  
 "And fired the shot heard round the world."

In that shot was stored the whole process of the liberation of man, including to-day's tragedy, as surely as the history of the oak, its growth, its majesty, and its fall, are stored in the acorn. The example of our freedom not only incited the more plastic peoples to democratization as in France and Britain, but also started a process of selection leading straight to this death grapple between democracy and autocracy. For by selection those most impatient of despotic control, those of greatest initiative, in short the most independent from all other lands, immigrated to us and to the other democracies induced by our example, leaving as a residuum the less independent, the more docile.

The democratizing impulse of our example, aided as it was in France by hunger and by outraged manhood, and in England by royal fatuity, was irresistible. But in Prussia it was so far weakened by material comfort created by the skill and craft of her kings that it failed, and in failing exaggerated this process of selection.

That failure found the Teutons without those influences which in

France and Great Britain temper the selection by emigration, the inborn domesticity of the French, and the closeness of the ties between Britain and her colonies, which leads great numbers of her sons to send their children back to school, and to return to the motherland later in life when their adventurousness has spent itself, thus restoring to the nation much of its more adventurous blood which the dreams of youth send abroad. Because Germany was intensely autocratic and without hope of democratization, she lost by emigration not only the most adventurous and independent, but even a vast number of those of intermediate independence, to our great gain. Thus the mean docility of the residuum was exaggerated. Because Germany lacked colonies these emigrants identified themselves with the countries to which they went, and renounced allegiance to their land of birth, so that the submissiveness of those left at home was undiluted by the return in later life of the more independent stock.

It is this exaggerated docility, resulting thus irresistibly from the selection started by our liberation, that has led to this terrible struggle. For it has enabled the Hohenzollerns and Hapsburgs, through their Nietzsches and their Treitschkes, through a cunningly devised state education appealing to all the baser motives, to ensnare the Teutons, drunk with vanity and new riches, in the intellectual slavery implied by their adopting the monstrous doctrine that might makes right, with all its sickening corollaries, its frightfulness, its foulness, and its treachery,

“hideous as the deeds

“Which you can scarce hide from men’s revolted eyes.”

It is this docility, with the resultant intellectual enslavement, that has enabled these autocrats to weld their people into a mass so coherent, so blindly unquestioning, as to strike with an initial force wholly disproportionate to their numbers when compared with those of democracies, whose units, however strong individually, cohere but feebly in everyday life because of their individualism.

It is this frightfulness, it is this depth of depravity and cruelty engrafted on all Teutonia through this same process, that has added incalculably to the danger that autocracy will win, by frightening into neutrality or even into ill-concealed coöperation Germany’s small neighbors who are our natural allies. Such a victory, leaving the Hohenzollerns and Hapsburgs still in power, could at best be followed by a truce between two armed camps, autocracy and democracy, during which the world would be putting all its available energies into arming for a renewal of fighting.

Slow as we have been to understand that this struggle is something more than the latest of the European wrangles “whose obscure sources do not interest us,” once we let our eyes be opened, once we saw that a great moral principle was at stake, we resolved that as it was we who had begun the extirpation of autocracy, so we must share in carrying this work to its triumphant end, not because any other course might well leave us to fight Teutonia singlehanded but because we are determined that “government of the people, by the people, for the people shall not perish from the earth.”

Our efforts should be quickened by the thoughts that though the means of creating men cannot in the nature of the case be hastened, those of destroying men can be accelerated to an indefinite degree by

the applications of science. The very qualities which have enabled man to better himself at a rapidly accelerating rate by his inventions, the steam engine, anesthesia, the sewing machine, the motor car, the telegraph, the telephone, and wireless telegraphy, are capable of being used to increase at an equally accelerating rate the means of slaying him. From the enormous increase in the powers of human destruction in the few decades since Sedan we get but a first foretaste of what man is inherently capable of inventing. We are indeed shocked at the new and higher order of destructiveness of this war, but our imagination fairly reels in trying to conceive the still higher order of ruin of future wars, should autocracy stay in the saddle. Those against whom this enhanced destruction, this annihilation, would be launched would not be the savage and uncivilized but the very flower of the race, those who threaten autocracy most. Because man cannot deprive his fellows of these powers of invention, he can prevent his own annihilation only by curbing autocracy, that form of government which evolves naturally through human destruction, or better by destroying it and replacing it with democracy, which evolves naturally through human betterment.

Our problem at the end of the war will be to prevent future wars, to prevent the actual employment of the enormously enhanced destructive powers sure to evolve, to force the nations to keep the peace as we have forced individuals. To say that we are inherently incapable of preventing our own annihilation; that because the less developed past did not learn to prevent its little wars, killing their tens of thousands, the more developed future must ever remain impotent to prevent its wars with their far higher order of havoc, killing their millions, is to betray a fatalism, a pessimism as unworthy of America as it is foreign to our nature. To me it seems an insult to the mothers who bore us, nay, to the God who made us.

## APPENDIX

### *The Record of Henry Marion Howe*

#### I. BIOGRAPHIC DATA

- Born March 2, 1848, at Boston, Mass.
- Graduated from Boston Latin School, 1865.
- Graduated as A. B. from Harvard College, 1869.
- Graduated (degree corresponding to B. Sc.) from Massachusetts Institute of Technology, 1871.
- Received degree of A. M. from Harvard College, 1872.
- Student in Steel Works at Troy, N. Y., 1871-1872.
- Superintendent of Bessemer works of Joliet Iron and Steel Co., 1872.
- Superintendent Blair Iron and Steel Co.'s Works, Pittsburgh, 1873-1874.
- Married Miss Fannie Gay, of Troy, N. Y., 1874.
- Improvements in copper smelting in Chile for heirs of Augustus Hemenway, 1877-1878.
- Designed and built the works of the Orford Nickel and Copper Co., at Capelton and Eustis, in the Province of Quebec, Canada, and at Bergen Point, N. J., 1879-1882.
- Manager of Pima Copper Mining and Smelting Co., Arizona, 1882.
- Consulting metallurgist with office at Boston, and Lecturer on Metallurgy at the Massachusetts Institute of Technology, 1883-1897.
- Professor of Metallurgy at Columbia University, New York, 1897-1913, and Professor Emeritus, 1913 to date.

## II. DISTINCTIONS RECEIVED

*Honorary Memberships*

Royal Swedish Academy of Sciences.  
 Russian Imperial Technical Society.  
 Russian Metallurgical Society.  
 Cleveland Institution of Engineers, England.  
 Société d'Encouragement pour L'Industrie Nationale, of France.  
 Dallas Historical Society, Texas.  
 Alumni Association of the School of Mines of Columbia University.  
 American Iron and Steel Institute.  
 Institution of Mining and Metallurgy, England.

*Fellowships, etc.*

National Academy of Science (Member).  
 New York Academy of Science (Fellow).  
 Washington Academy of Science (Non-Resident Member).  
 American Academy of Arts and Sciences (Fellow).  
 American Philosophical Society (Non-Resident Member).  
 American Academy of Engineers (Charter Member).

*Presidency*

American Society for Testing Materials (four terms 1900-1902, 1909-1912).  
 American Institute of Mining Engineers, 1893.  
 International Association for Testing Materials, 1912.  
 Alumni Association, Massachusetts Institute of Technology (three terms).  
 Jury of Awards, Mining and Metallurgy, Chicago Exposition of 1893.

*Honorary Vice-Presidency*

Iron and Steel Institute of Great Britain.

*Life Membership of Council*

International Association for Testing Materials.

*Orders*

Knight of the Order of St. Stanislas, Russia (second order, with star of first order).  
 Chevalier of the Legion of Honor of France.

*Medals*

Bessemer Medal, Iron and Steel Institute of Great Britain.  
 Eliot Cresson Medal, Franklin Institute of Philadelphia.  
 Gold Medal of the Verein zur Befoerderung des Gewerbflusses, Berlin.  
 Gold Medal of Société d'Encouragement pour l'Industrie Nationale of France (1916).

*Honorary Degrees*

LL.D., Harvard, 1905.  
 LL.D., Lafayette, 1905.  
 Sc.D., University of Pittsburg, 1915.

## III. PUBLICATIONS

In this incomplete list of writings of Henry M. Howe the following abbreviations are used:

A. I. M. E.—Transactions of the American Institute of Mining Engineers.  
 A. S. M. E.—Transactions of American Society of Mechanical Engineers.  
 A. S. T. M.—Proceedings of the American Society for Testing Materials.  
 I. A. T. M.—Proceedings of the International Association for Testing Materials.  
 J. I. S. I.—Journal of the Iron and Steel Institute.  
 E. & M. J.—Engineering and Mining Journal.

*Principal Works*

The Metallurgy of Steel, Scientific Publishing Co., New York, 1891, translated into French.

Metallurgical Laboratory Notes, 1902, translated into French.

Iron, Steel, and Other Alloys, 1903, translated into Russian.

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Article on Iron and Steel in the New Volumes of the 10th Edition of the Encyclopædia Britannica, 1902.

Article on Iron and Steel in the 11th Edition of the Encyclopædia Britannica, Volume 14, page 801, 1910.

*Professional Papers*

Improvement in Processes and Furnaces for Reducing and Smelting Ores; one-half assigned to Mr. Howe by William E. C. Eustis, U. S. Patent No. 209,554, November, 1878.

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A Direct Process of Copper Smelting, A. I. M. E., 7, 1878-79.

Two New Processes for the Extraction of Nickel from its Ores, A. I. M. E., 9, 1880-81.

On Comparative Efficiency of Fans and Positive Blowers, A. I. M. E., 10, 1881-82.

Contributions to the Metallurgy of Nickel and Copper, A. I. M. E., 10, 1881-82.

A Suggested Cure for Blast-Furnace Chills, A. I. M. E., 11, 1882-83.

A Systematic Nomenclature for Minerals, A. I. M. E., 12, 1883-84.

The Patience of Copper and Silver as Affected by Annealing, A. I. M. E., 13, 1884-85.

Note on the Contraction of Iron under Sudden Cooling, A. I. M. E., 14, 1885-86.

Attainment of Uniformity in the Bessemer Process, A. I. M. E., 15, 1886-87.

Smelting Cupreous Pyrites, E. & M. J., March, 1879.

Bessemerizing Sulphides, E. & M. J., May, 1879.

Bessemerizing Matte in a Reverberatory Furnace, E. & M. J., March, 1883.

Bessemerizing Copper Matte, E. & M. J., April, 1883.

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The Hunt and Douglas Copper Process, E. & M. J., December, 1885.

Rose Polytechnic Inst., Annual Scientific Address, 1885.

Copper Smelting, Bulletin 26, U. S. Geol. Survey, 1885.

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The Attainment of Uniformity in the Bessemer Process, E. & M. J., June, 1886.

The Effect of Slag on the Fibrousness of Avesta Steel, E. & M. J., September, 1886.

The Elizabeth Copper Mine, Vermont, E. & M. J., November, 1886.

The Manufacture and Cost of Coke, E. & M. J., November, 1886.

The Quality of Steel for Guns, E. & M. J., January, 1887.

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Two Conditions of Phosphorus in Iron, A. I. M. E., 16, 1887.

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Why do Steel-Tired Wheels Wear Flat Less than Chilled Cast-Iron Ones? E. & M. J., July, 1891.

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 The Physics of Steel, A. I. M. E., **24**, 1894.  
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 The Relative Corrosion of Wrought Iron and Steel, The Mineral Industry, 1895.  
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 A Possible Explanation of Kernel Roasting, E. & M. J., March, 1895.  
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 The Relation Between Temperature and the Grain of Steel, by A. Sauveur and H. M. Howe, E. & M. J., December, 1895.  
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 Progrès Réalisé depuis 1889 dans la Metallurgie du Fer et de l'Acier, Bulletin de la Société de l'Industrie Minérale, 1901.  
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 Metallurgical Laboratories, Science, n. s., May 16, 1902.  
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 The Relative Corrosion of Steel and Wrought-Iron Tubing, A. S. T. M., September, 1908.  
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 Influence of Top-Lag on the Depth of the Pipe in Steel Ingots, A. I. M. E., **40**, 1909.  
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Address at the Formal Opening of the New Buildings of the Perkins Inst. and Mass. School for the Blind, June 4, 1914.  
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Notes on the Hardening and Tempering of Eutectoid Carbon Steel, A. S. T. M., 16, No. 2, 1916.  
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Recrystallization after Plastic Deformation, A. I. M. E., *Bulletin*, October, 1916.  
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## INTERNATIONAL MINING CONVENTION, NELSON, B. C.

The International Mining Convention held in Nelson, B. C., May 17 to 19, was attended by nearly 500 delegates, of whom about 100 were from Spokane, 150 from the Slocan district, and the others from different points in Canada and the United States.

Arrangements were in charge of the Nelson, Slocan and Eastern British Columbia Mining Men's Club, under the direction of Secretary Fred A. Starkey, assisted by the City of Nelson, Associated Boards of Trade of Eastern British Columbia and Nelson Board of Trade. The Columbia Section of the American Institute of Mining Engineers, cooperating with the Western Branch of the Canadian Mining Institute, provided the technical program, under the management of Secretary E. Jacobs, of Victoria, and Secretary L. K. Armstrong, of Spokane.

Three technical sessions were held, at which the following papers were read:

- Difficulties of the Electrolytic Zinc Process, by Thomas French.
- Sketch of the Geology and Ore Deposits of Nelson and Kootenay Region, by Arthur Lakes.
- Advantages of Local Organization of National Societies, by W. H. Linney.
- Review of Kootenay Mining, by Bruce White.
- Milling of Partially Oxidized Ores, by S. S. Fowler.

## Resolutions

The Resolutions Committee presented the following resolutions, which were adopted unanimously, after discussion:

*The Coal Situation*

"Whereas, The serious condition created by the strike in western Canada is having a most disastrous effect upon all industries, which will shortly be paralyzed:

"Resolved, That the Dominion government be asked to take immediate temporary possession of all coal mines now idle, or which may be idle, and operate them to their full capacity to produce coal and coke; that a commission be appointed at once with full power to take evidence on oath, and that full investigation shall be made as to who is in the wrong in the dispute between capital and labor, and whoever it may be, that the wrongdoers shall be compelled to do their duty, if necessary, under the application of military law. This convention takes no attitude on the dispute between capital and labor, but it believes that the prosperity of the whole Dominion, and possibly the success of the Allies in the prosecution of the war, transcends in importance a dispute that should be settled by a power greater than either labor or capital."

*Mining Law Changes*

"Whereas, the provincial government has brought in a bill containing radical changes in the mining laws; Resolved, That this convention believes that any changes at this time would tend to discourage investment of capital and that, before making any changes at any time, the responsible officials of the government should consult fully with miners and mine owners' associations of British Columbia to the end that the knowledge thus gained by practical experience be incorporated in such amendments."

*French Electrolytic Process*

"Whereas, Following an address by Thomas French concerning the French electrolytic process, this convention is convinced that in such a process the country possesses a potential factor for economical recovery of vast reserves of zinc in complex ores; Resolved, That this convention recommends that the Dominion government thoroughly investigate this process through the best technical knowledge available, and that if it be found practical, it be aided and protected by government subsidy, to the end that these great resources of the Dominion be developed forthwith."

*Favors Selective Conscription*

Another resolution commended the Dominion government's action in modifying an order-in-council prohibiting aliens from holding property in Canada to permit of uninterrupted investment by people of friendly alien nations.

A resolution sent up from the floor of the convention by A. W. Watts, of Wattsburg, and passed with a cheer, endorsed suggested legislation in favor of selective conscription, as promised in press dispatches of that day by Premier Borden.

*The Banquet*

The social features of the Convention were thoroughly enjoyed by all who took part in them, especially the banquet in honor of the visiting delegates. The toasts and responses at this banquet were full of patriotic enthusiasm and of the spirit of brotherhood existing and growing between Canada and the United States, the part that each country is taking in the war forming the main theme of a greater number of the speeches.

Sidney Norman, of Spokane, who was toastmaster, in his expression of appreciation for the honor conferred upon him, struck the keynote of the evening. He said:

"When I come to Nelson or any part of West Kootenay, I am by no means a stranger. It is twenty-four years and three months since I first came down the Kootenay River.

"The love I learned for the people of this country has been with me since those days, and it has been immeasurably increased by the happenings of the past three

years. I wonder if you who came with me today upon a holiday trip have any conception of what has been going on in Nelson, as in every other town of the great British Empire, upon which the sun never sets? I wonder if you know what war means?"

Mr. Norman then spoke of the great work that has been done by the people of the Kootenay district during the war, and ended by proposing the first toast of the evening:

"The toast is to him who represents one of the two greatest democracies of history; who represents the greatest civilizing agency the world has ever known—the well-spring of the great basic laws of freedom upon which our own nation is founded. He is a sovereign, 'tis true, but he is not a Kaiser and he is not a Czar—(Applause). He is just a man, like you, like me; with heart that beats for country, wife and children; a man of great responsibility, without menacing power; a gentleman, the first gentleman in the British Empire—Ladies and gentlemen, 'THE KING.' (Cheers and great applause.)

### *To the President of the United States*

A toast to the President of the United States was then proposed, by William Hunter, a Canadian, and was followed by the singing of "The Star Spangled Banner."

### *Toast to the Institute*

The third toast, to the American Institute of Mining Engineers, was responded to by Mr. Philip N. Moore, President of the Institute. He also laid emphasis upon the tie between the two great countries, Canada and the United States, in saying:

"I come to you with a greeting from the great kindred society across that imaginary line which divides our lands—that wonderful line where for three thousand miles and for one hundred years no visible indication of possibility of hostilities has come between us. I come to you with the greetings of that great society, with dreams of what it may be some day, which I cannot burden you with now, but the hope of the realization of those dreams comes nearer home as I see the enthusiasm of the men of our own organization and as I meet the men of our profession on this side of the line. I can see coming one great national organization of engineers in our land, ultimately numbering fifty, possibly a hundred, thousand members, representing all the branches of the profession. And is it too much of a dream that I may hope you, in your turn, may be of those numbers? (Applause.)

"I come here tonight to find hanging over you the spirit of sober things. You who have been known to be heroes for so many months, who have given of your best and your dearest to the cause of civilization, may feel it in ill time in us, who are just beginning, to tell stories and ancient tales. But, gentlemen, with us this spirit has been working, and that lonely man in the White House, one of whose critics I frankly acknowledge I have been—one of whose severest critics—has shown us that in that apparent delay and indecision there has been a method which has brought about a condition wherein he has back of him, as never before, a united public opinion throughout our whole land. We have reproached ourselves with our materialism; our neighbors both on this side of the water and across it, have not been lenient in the same regard. Perhaps it is true—God knows some of us are materialists—perhaps the great trial ahead of us may be needed to bring us back once more to a realization that there are better things in life than making money and worse things than death. (Applause.) And so, gradually bringing up to him the united voice of public opinion, our leader goes forward slowly and hesitatingly, it may seem, but I can say to you that the great nation across the line is just beginning to feel the task ahead of it. But the task once undertaken, gentlemen, is surely done. (Applause.)

"And for you of Canada, what message can I bring? Your message was written for you years ago. What is it?

" 'Carry the word to my sisters,  
To the queens of the East and the South,  
That I have proven a heritage  
By more than word of mouth.

They that are wise shall follow  
 When the world's war trumpet blows,  
 But I, I am first in battle,  
 Said our Lady of the Snows.'

"Then back across the stormy waters comes the answer:

Tempered, august, abiding,  
 Reluctant of prayers and vows,  
 Eager in face of peril,  
 As thine for thy mother's house;  
 God requite thee, my sister,  
 In the wonderful years to be,  
 And may thy people too, love thee  
 As thou hast loved me."

(Great Applause).

### "The Allies"

In proposing the toast to "The Allies," Mr. Norman spoke of the admiration of all people for the heroes of the European battlefields, of the sympathy and sorrow for those who have given their all, and of the hope that the boundary line between the two great countries on this continent, "always unreal, has been wiped out forever." He said:

"We enter this struggle 'with malice toward none, with charity for all, with firmness in the right as God gives us to see the right. Let us strive on to finish the work we are in, to bind up the nation's wounds; to care for him who shall have borne the battle and for his widow and for his orphan, to do all which may achieve and cherish a just and lasting peace among ourselves, and with all nations.' Ladies and gentlemen, this is the toast:

"The Allies, a Quick War Alliance and an Eternal Alliance in Peace.'"

For the United States, the response to this toast was made by Judge George Turner, of Spokane. He spoke of the cordiality always shown by the residents of British Columbia to their "cousins" from across the border, and said:

"There is perhaps a reason for this obliteration of the border line between us, which may not be found in the relations of other nations—a reason that has peculiar significance at this time and is therefore worthy of a little consideration. We are practically one people, one in blood, in language, in law, in literature, and in the essentials of the free government under which we live and have our being. For all these things we go back to a common origin. Border lines between such peoples are essential, no doubt, for the purposes of orderly government, but, for the purposes of social and business intercourse, a comity that makes them as nearly invisible as is possible is the natural and ideal condition.

"And this brings me, Mr. Toastmaster, to the titanic struggle that is now convulsing the world, in which all the branches of the English people are waging gallant battle in the cause of freedom and humanity.

"The people of the United States, the other branch of the English-speaking people on the American continent, have taken their place beside their English brethren, and thus all the English-speaking people of the world are in arms, standing shoulder to shoulder in defence of English ideals of right and justice and liberty; in defense of the right of man the world over to live his life in his own way, without fear of molestation, and under institutions of his own free choice.

"I shall not exalt the courage and valor of the American soldier, or the strength and power of the American Republic. The first has never been found wanting. The second has never been exerted except in a just cause, and, its standard once raised, has never been lowered until its cause was fully and completely vindicated."

The reply for the British Empire was made by Judge Forin, of West Kootenay, Commanding Officer of the 107th Regiment. Confirming Judge Turner's statement regarding the friendly relations between the United States and Canada, he said:

"Why! the American nation is nothing more than the elder son of our race, that is all. He left home before 'The Lady of the Snows' was written. He left before we had learned the lesson that a nation can be a daughter in her mother's house, but mistress of her own. (Applause.) When a brother enters into war with a brother, when a nation divides, when a nation breaks away from its parent, as the American Republic did, and justifies its action, it does not follow that families shall be eternally divided. As your great Daniel Webster said fifty years after the Republic was established, 'We have both profited by it.' I believe that the separation of the United States at that particular time, for the principles which they stood for, the principles of freedom, for an ideal, directed by George Washington and the other British officers and citizens, was a blessing to our race—the Anglo Saxon race.

"In closing, because I see many friends about here, I want to offer a word of sympathy. While we are here enjoying ourselves, sitting around the festive board, which is part of the day's work (and the boys in the trenches would not have it otherwise), we cannot forget that they are in the front of one of the most desperate battles ever endured. And in looking around I see here families who have given so much. You have given, of course, what you value greater than your own lives. Now, you know I am expressing the sympathy of all those within hearing. We don't forget you—we will not forget you. And as for Corporal Berry, the returned soldier who has spoken, he need not fear for the time to come, that time when the boys who have served and are coming back—we are going to take care of them—we have got to take care of them. (Applause.)

"Let me give you a second verse for God Save the King, and take it home with you to the American side. It is:

"God save our splendid men,  
Bring them safe home again,  
God save our men.  
Make them Victorious,  
Patient and Chivalrous,  
They are so dear to us,  
God save our men."

(Applause.)

Reverend Captain Hindley spoke for the Republic of France, mentioning also the gallant work that has been done and is being done now by all the allied nations. In spite of the "blot on the 'scutcheon'" of Ireland, placed there by plotters and excitable men, the Irish regiments in France, he said, share the glory and the honor with the English Tommy and the Scottish Highlander, with the Colonial from Australia, from New Zealand, from South Africa, or from Canada. He ended his speech with a special tribute to France, and a prophecy that the spirit of brotherhood and democracy will be strengthened by the work of the allied nations for the common good.

## PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members)

The following is a partial list of members and guests who called at Institute headquarters during the period May 10, 1917, to June 10, 1917.

F. J. Anderson, Cali, Colombia.  
 George J. Bancroft, Denver, Colo.  
 W. F. B. Berger, Denver, Colo.  
 F. J. Brulé, Toronto, Ont., Canada.  
 J. Murray Clark, Toronto, Ont., Canada.  
 Will L. Clark, Jerome, Ariz.  
 G. H. Clevenger, Palo Alto, Cal.  
 Paul S. Couldrey, Sound Beach, Conn.  
 C. W. Crispell, Kingston, N. Y.  
 George A. Easley, La Paz, Bolivia.  
 Bancroft Gore, Butte, Mont.  
 John B. Guernsey, Roanoke, Va.  
 Rush M. Hess, St. Louis, Mo.  
 E. Hibbert, Nickelton, Ont., Canada.

W. F. Jahn, Kirkland Lake, Ont., Canada.  
 W. McA. Johnson, Hartford, Conn.  
 J. E. Jopling, Marquette, Mich.  
 William Kelley, Vulcan, Mich.  
 H. M. LaFollette, La Follette, Tenn.  
 Dorsey A. Lyon, Washington, D. C.  
 James F. Magee, Lancaster, Pa.  
 M. Ortin, Nijni-Novgorod, Russia.  
 C. H. Palmer, Jr., Los Angeles, Cal.  
 Edison S. Pettis, Mill Valley, Cal.  
 Mortimer A. Sears, Paintsville, Ky.  
 W. J. Uren, Houghton, Mich.  
 C. E. Visel, New Haven, Conn.

Percy E. Barbour, has resigned his position as managing editor of the *Engineering & Mining Journal*, to accept the appointment of Deputy Superintendent of the New York State Troopers, with headquarters at Albany, N. Y., with the rank of captain. He has also been commissioned as captain in the 22nd Regiment, N. Y. N. G., having been formerly 1st Lieutenant.

J. R. Buchanan is in charge of the mining operations of the Hanover Bessemer Iron and Copper Co., Fierro, N. M.

Forest B. Caldwell has been elected a director and is consulting engineer for the Morning Glory Co., San Francisco.

G. Howell Clevenger has been appointed research professor of metallurgy at the Leland Stanford University, Palo Alto, Cal.

Will H. Coghill has been appointed consulting metallurgist in the coöperative work of the Oregon Bureau of Mines and Geology and the U. S. Bureau of Mines.

S. H. Eu has been appointed manager of the Kadaik Mining Syndicate L.C.T., the first tin producing district in Burmah.

George E. Farish has been appointed manager of the National Leasing Co., of Nevada.

Charles R. Fettke is examining oil lands in Allen, Carter and Montgomery Counties, Kentucky.

Carroll R. Forbes, professor of mining of the School of Mines, University of Missouri, is a captain in the Engineer Officers' Reserve Corps. He is at present at Ft. Riley, Kan., but expects to go shortly to Ft. Leavenworth.

C. E. Grunsky, Jr., Secretary-Treasurer of the San Francisco Section, is a Captain in the 8th Reserve Engineer Regiment, of San Francisco.

Joseph H. Hill is employed in the Engineering Department of the Nevada Consolidated Copper Co., Ruth, Nev.

**B. Hochschild** was elected chairman of the board of directors of the American Metal Co. at a recent meeting.

**Julian Kennedy**, consulting engineer of Pittsburgh, Pa., has been appointed a member of the State committee of public safety by the Governor of Pennsylvania.

**M. H. Kuryla** has received an appointment with the U. S. Smelting Co., Silverton, Colo.

**Montrose L. Lee**, who has been prospecting for S. Pearson & Son, Ltd., in Hayti has returned to New York and may be reached at 65 Broadway.

**W. H. Leonard**, president of the Denver Rock Drill Co., has organized and been elected captain of Troop E, First Regiment, Colorado Cavalry.

**Robert S. Lewis**, professor of mining engineering of the University of Utah, has returned to Salt Lake City after examining a mine in Arizona. He will spend the summer at Huntington Lake, Cal.

**C. M. Loeb** was elected president of the American Metal Co.

**Frederick Maccoy** is mining engineer for the Sunnyside Mining & Milling Co., Eureka, Colo.

**Bernard Macdonald** has gone to Chile as consulting engineer. He expects to be there for some months. His address is Casilla No. 20, Antofagasta, Chile.

**Charles H. Macdowell**, Chairman of the Chicago Local Section, is a member of the Committee on Chemicals of the Advisory Committee of the Council of National Defense.

**Sydney A. Mewhirter** is Designate Lieutenant of the Engineer Officers' Reserve Corps, U. S. A., at Ft. Leavenworth, Kansas.

**Seeley W. Mudd** is a Major in the 8th Reserve Engineer Regiment of San Francisco.

**W. H. Nichols** is Chairman of the Committee on Chemicals of the Advisory Committee of the Council of National Defense.

**H. J. Rahilly**, mining engineer, U. S. Bureau of Mines, has resigned to accept a position with the Anaconda Copper Mining Co., Butte.

**Stuart L. Rawlings** has been appointed assistant general manager of the Cerro de Pasco Copper Corporation, Lima, Peru.

**Clyde P. Ross** has enrolled in the Harvard Regiment of the Reserve Officers' Training Corps.

**Herbert W. Ross** has resigned as assistant manager of the Backus & Johnston Co., of Casapalca, Peru, and is now located at 314 Pacific Ave., Piedmont, Oakland, Cal.

**Francis S. Schimerka** has been appointed Research Engineer for the milling department of the Arizona Copper Co., Ltd., at Clifton and Morenci, Arizona.

**S. F. Shaw**, superintendent of the Tiro General mine of the American Smelting and Refining Co., is reopening the Panuco mine near Monclova, Coah., for the same company.

**William A. Sloan** is employed as chemist and assayer for the United Verde Extension Mining Co., Jerome, Ariz.

**George Otis Smith**, director of the United States Geological Survey,

has been appointed a member of the National Research Council by request of the President.

**Harrison Souder**, formerly general superintendent of the Cornwall Ore Bank Co., has been appointed general manager of the company.

**E. Gybbon Spilsbury** sailed for Central America on June 5, 1917, and expects to be absent until the middle of July.

**H. N. Stronck** has been appointed vice president and managing engineer of the Illinois Engineering, Auditing & Public Relations Co., 122 So. Michigan Ave., Chicago, Ill., this firm being successors to Benedict, Boyle & Stronck, Inc., and is engaged in general consulting business on engineering, construction, industrial economics, public relations, accounting and auditing.

**Henry N. Thomson** has resigned as metallurgist for the United Verde Copper Co., and is engaged in consulting practice at 918 South Kingsley Drive, Los Angeles, Cal.

**Benjamin W. Vallat**, formerly superintendent for the Newport Mining Co., and now president of the Dominion Forge and Stamping Co., has been appointed major of a battalion of engineers to be organized at the Washington College of Mines.

**C. G. Visel** has resigned the superintendency of the Mile Wide Copper Co.'s property.

**William H. Wagner** has been appointed general manager of the Copper King Mines Products Co., with headquarters at Copperton, Eagle Co., Colo.

**Arthur T. Ward**, experimental engineer for the Braden Copper Co., Rancagua, Chile, has returned to this country and is now at Bellefonte, Pa.

**John H. Winchell, Jr.** is engaged as engineer for the Demming Mines Co., Murphy, Idaho.

**Earnest Wittenau** has been appointed assistant superintendent of concentration for the Arizona Copper Co., Ltd., at Morenci.

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## POSITIONS VACANT

No. 219. Assayer for gold-silver property in Ecuador.

No. 220. Two engineers to go to Russia, 22-28 years old. Not engaged or married. Knowledge of some language other than English. To study conditions in Russia two summers and intervening winter. One man must have experience in oil-well operation or oil manufacture; the other, in mining or metallurgy. Salary £300 per annum, plus traveling and living expenses.

No. 221. Position in metallurgical testing department. Initial salary \$1600 to \$1800. Candidate who has had industrial training and experience either in the manufacture and testing of steel or in electro-metallurgy preferred, but one who has had post graduate work in these lines will be considered.

No. 223. Experimental and research laboratory in Wisconsin wants a man who has had experience in ore dressing, iron ore preferably. Salary \$3600. Also, an iron metallurgist. Salary \$3600.



No. 224. Superintendent for bituminous coal mine in Pennsylvania. Energetic man who has had experience in Pennsylvania soft coal fields and who is able to handle men. Slope mine now being dewatered. Will start production at 500 tons and will increase shortly to 1,000 tons. Salary \$200-\$250 a month.

### ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members)

Manager or general superintendent. Twenty years experience mining and milling, familiar with all stoping methods. Will guarantee efficiency and low cost, past record, good, fluent Spanish, best of references. No. 361.

Concentrator superintendent. Just returned from abroad. Open for engagement. Exceptional experience designing, construction and operation. References A1. Twenty years experience. No. 362.

Member, also Assoc. Mem. Am. Soc. of C. E.; excellent technical education, and best of references covering twelve years in responsible charge of work; experienced in surveys, construction, prospecting, mine examination and development, and irrigation work; speak Spanish fluently, and can write a good report. Good executive. Age 34; married. No. 357.

Member in Antofagasta, Chile, wishes commissions for private professional work in making examinations of mines and in designing mine and mill equipment in the tributary mining regions. No. 363.

Member, age 38, thirteen years' general experience, six in government service on geological work in eight western States, wishes position with mining or industrial company as geologist or engineer. No. 364.

Member, graduate mining engineer, desires position as mine superintendent or assistant superintendent. Sixteen years' experience in mill machine shop, shift boss, chief engineer and superintendent in copper and iron mines. Age 37. Married. Location in town with good schools desired. Open for engagement after July 1. References. No. 365.

Member, technical mining graduate, fifteen years' experience in mining and milling as engineer, superintendent of mines and mills, examining engineer, etc. in the United States and South America. Can furnish good references and desires position with mining company that can do likewise. No. 366.

### LOCAL SECTION NEWS

#### CHICAGO SECTION

C. H. MACDOWELL, *Chairman*,  
G. P. HULST, *Vice-Chairman*,

H. W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History, Chicago, Ill.

R. S. BONSB,  
HORACE H. CLARK,

H. B. PULSIFER,  
H. T. WALSH.

Seventy-four members of the two organizations attended a three-day joint field meeting of the Chicago Section and the Illinois Mining Institute at La Salle, Ill., on May 17, 18 and 19.

The meeting began with a technical session in the rooms of the Illinois

Valley Manufacturers' Club on Thursday afternoon. After an address of welcome by the Mayor of La Salle, Dr. H. M. Orr, and response by Fred S. Pfahler, President of the Illinois Mining Institute, the following technical papers were presented: Roller Bearing Trucks, by E. B. Mead; Boiler Plant Efficiency, by Fred Laufketter, followed by a prolonged discussion led by E. E. Billow; Storage Battery Motors by Mr. Hall of Ironton, Ohio.

On Thursday evening, the members and friends attended a reception in the rooms of the Illinois Valley Manufacturers' Club.

The morning of Friday was taken up by another technical session at the Kaskaskia Hotel. The session was opened by a paper on the History of Longwall Mining by E. T. Bent, President of the Illinois Coal Operators Association. A discussion of this paper soon turned into a symposium on the differences in the character of the surface subsidence consequent on room-and-pillar and on longwall mining. The experience of the members present had been so varied that this unexpected discussion proved to be the most interesting and valuable technical feature of the meeting.

On Friday afternoon, the members first inspected the works of the Western Clock Co., where Big Ben is made, and then separated to take the optional trips to the plants of the Illinois Zinc Co., The Matthiessen and Hegeler Zinc Co., the La Salle County Carbon Coal Co. and the Silica Mines at Ottawa. The majority of the members chose to view the Matthiessen and Hegeler plant, where all stages of zinc smelting and rolling were seen, from the freight-car unloaders to the shipment of finished sheet zinc.

On Friday evening there was a dinner in the grill room of the Kaskaskia, followed by a talk by E. C. Lee on Merit Rating of Workmen's Compensation Insurance. This was followed by the Section's annual business meeting, which elected officers for the year 1917-18.

The later evening was occupied by an open meeting in the High School Auditorium, where the State Geologist, F. H. DeWolf, talked on The Geology of the Longwall Region.

Dr. DeWolf was preceded by C. H. Modderwell, who told us in a very interesting manner of the work that one of the Sections' members, F. S. Peabody, is doing as the head of a fuel committee of the National Defense Council to insure the proper utilization, distribution and conservation of the Nation's fuel during the war.

Saturday morning the party was transported by automobile to the Vermillion River Bridge where groups were formed for the optional excursions: A foot tour along the La Salle anticline, which is exposed in a canyon of great scenic beauty in Deer Park; inspection of the plants of the Lehigh Portland and The Marquette Cement companies; The Black Hollow Coal Mine of the Illinois Zinc Co. and the Jones Fire Clay deposit. The Black Hollow Mine showed, in successful use, a system of longwall mining on a steep pitch and other interesting features due to mining on the steep flank of the anticline. At noon there was a luncheon at the Deer Park Country Club. During luncheon, the Secretaries of the two organizations were instructed to transmit to Mr. Peabody the following resolution, which was unanimously adopted:

The Chicago Section of the American Institute of Mining Engineers and the Illinois Mining Institute at their joint meeting in La Salle, after listening with great interest to Mr. C. H. Modderwell's account of the

work of Mr. F. S. Peabody and his Committee of the National Defense Council, have

**RESOLVED:** That the members of the Chicago Section of the American Institute of Mining Engineers and of the Illinois Mining Institute present at their joint meeting in LaSalle will assist in every possible way the work of Mr. F. S. Peabody and his Committee of the National Defense Council for the better conservation and utilization of the fuel resources of the Country.

On Saturday afternoon the members went by automobile to Starved Rock Park, returning to La Salle in time to take the late afternoon trains to Chicago.

HENRY W. NICHOLS, *Secretary*.

### NEVADA SECTION

J. W. HUTCHINSON, *Chairman*,

FRANCIS CHURCH LINCOLN, *Vice-Chairman*.

HENRY M. RIVES, *Secretary-Treasurer*, Reno, Nevada.

W. H. BLACKBURN,

E. A. JULIAN,

EMMET D. BOYLE,

JOHN G. KIRCHEN,

FREDERICK BRADSHAW,

C. B. LAKENAN,

TASKER L. ODDIE.

A special meeting of the Nevada Section was held in Reno on May 3, for the purpose of meeting President Moore and Secretary Stoughton of the Institute, who had arrived the previous night. Members were present from Tonopah, Goldfield, Aurora, Carson, Virginia City and Reno.

Mr. Stoughton in a most interesting address told of the coöperation of the National Engineering Societies with the Federal Government in the matter of national defense, with particular reference to the American Institute of Mining Engineers. The work of the local sections was then discussed, and Mr. Moore delivered a short address on matters pertaining to the Institute, which included among other things the question of raising the standard of admission. A general discussion thereupon ensued, after which the meeting adjourned.

In the evening a dinner was tendered to Mr. Moore and Mr. Stoughton by the members of the Nevada Section.

The next morning, the President and Secretary addressed the engineering students at the University of Nevada, later meeting the members of the student society affiliated with the Institute.

The annual field meeting of the Nevada Section is scheduled for June 22 and 23 at Ely, Nev.

### COLORADO SECTION

CHARLES LOUGHRIDGE, *Chairman*,

GEORGE M. TAYLOR, *Vice Chairman*,

FRED CARROLL, *Secretary-Treasurer*, State Capitol, Denver, Colo.

F. H. BOSTWICK,

P. M. McHUGH.

About sixty members of the local section of the Institute were present to enjoy the evening of May 25, with President Philip N. Moore. We were also fortunate in having with us at the same time, Mr. T. A. Rickard, of San Francisco.

The remarks of Mr. Moore were much appreciated, and general satisfaction was expressed by those present.

Mr. George M. Taylor, Manager for the mills of The Portland Gold Mining Co., gave some very interesting data on the operation of cyanide mills under the present abnormal conditions.

Thomas F. Stearns performed the part usually assigned to him, and came forward spontaneously with original and very appropriate stories.

This being Mr. Rickard's former home, many of his old-time friends were highly pleased to have the opportunity to again greet him and listen to a few timely remarks.

The meeting was well attended, considering the fact that this is a busy time for the Colorado mining men and the additional uncertainty due to the mobilization plans for war.

FRED CARROLL, *Secretary*.

### PUGET SOUND SECTION

SIMON H. ASH, *Chairman*,  
I. F. LAUCKS, *Vice-Chairman*,

CHARLES SIMENSTAD, *Secretary-Treasurer*, 425 Lyon Bldg., Seattle, Wash.  
GLENNVILLE A. COLLINS, JOHN N. POTT.

Mr. Philip N. Moore, President, and Mr. Bradley Stoughton, Secretary, visited Puget Sound Section on May 9 and 10, 1917. The visitors arrived at 8 p. m. on May 9. They were met by the entertainment committee and taken on a short automobile ride about the scenic highways of the city. During the forenoon of the following day visits were made to a few of the leading industrial plants on Elliott Bay and at noon an open-house luncheon was given at the Arctic Club.

The program was concluded with a banquet-reception given in honor of the visitors at the Arctic Club, at 6 p. m. that evening.

Thirty-five members and fifteen guests were present.

Dinner having been served, the meeting was called to order by the Vice-Chairman, I. F. Laucks, who, after an address of welcome to the visitors and guests, introduced Mr. F. C. Greene, the toastmaster of the evening.

Speeches and addresses were made in the following order: A. J. Rhodes, President of the Seattle Chamber of Commerce and Commercial Club, "Co-operation."

Rev. M. A. Matthews, Pastor of First Presbyterian Church of Seattle, "The Twenty-eighth Chapter of Job."

O. D. Fisher, General Manager of the Fisher Flouring Mills, "Resources of Our Country."

E. S. Brooks, Tono, Washington, "Washington Coal."

O. P. M. Goss, Seattle, Washington, "Lumber Products."

Colonel W. T. Perkins, Washington, Seattle, "Mines and Mining."

Mr. Ryan, U. S. Bureau of Mines, "Progress of the Seattle Station."

Mr. Stoughton spoke on topics of interest to the Institute, and asked for expressions on the subject of advertising in the monthly Bulletin, and on the subject of raising the requirements for admission to membership in the Institute. During the discussions that followed, all the members who expressed themselves were in favor of accepting advertisements for the Bulletin and of raising the requirements for membership.

Mr. Moore, the President, spoke on the work of the Institute on a National scale, and closed his remarks with a short talk on the "Belgian Kiddies, Ltd."

Following Mr. Moore's speech the following members subscribed to the "Belgian Kiddies, Ltd.": Mrs. Lena Allen Stoiber-Rood, G. M. Kerr, F. A. Hill, James A. Kelley, G. E. Rockefeller, F. Powell, B. N. Rickard, Willard V. Morse, W. H. Seagrave, W. T. Perkins, Roy Cooper, F. C. Greene, Percy E. Wright, Milnor Roberts, I. F. Laucks, L. W. Shirley, Chester F. Lee. The subscriptions amounted to \$1,217.

At the close of the discussions that followed, the reading of minutes of previous meeting and other business having been suspended, the meeting adjourned.

A business meeting of the Puget Sound Section was held at the Chamber of Commerce headquarters, May 15.

Chairman S. A. Ash being absent, Mr. Laucks presided.

A communication from the Associated Engineering Societies advising that the *Transactions* of the American Institute of Mining Engineers have arrived, and requesting that the local section make provision for shelving the same, was read. It was thereupon moved and carried that the associated body arrange for the shelving in a manner approved by our representative in the joint council.

The meeting was adjourned until May 21, 1917.

An adjourned business meeting of the Puget Sound Section was held at the Arctic Club on May 21.

Five members were present.

Vice-Chairman I. F. Laucks presided.

Minutes of previous two meetings were read and approved.

It was regularly moved and carried that the chairman appoint two alternates to the joint council of the Associated Engineering Society.

There being no further business the meeting adjourned.

S. H. Ash, Chairman, in a communication to the secretary has appointed as nominating committee Messrs. George Watkin Evans, Percy E. Wright and Harry Boyle, and as the two alternates to the joint council of Associated Engineering Society, Messrs. Milner Roberts and Frank C. Greene.

CHAS. SIMENSTAD, *Secretary*.

## FORTHCOMING MEETINGS OF SOCIETIES

| Organization                                    | Place               | Date<br>1917 |
|---|---------------------|--------------|
| Mine Inspectors Institute of U. S. A. ....      | Indianapolis, Ind.  | July 9-11    |
| American Society of Sanitary Engineers.....     | Grand Rapids, Mich. | Aug. 1-3     |
| National Association of Stationary Engineers... | Evansville, Ind.    | Sept. 10-15  |
| National Exposition of Safety and Sanitation... | New York City       | Sept. 10-15  |
| American Iron, Steel & Electrical Engineers...  | Philadelphia, Pa.   | Sept. 10-14  |
| American Institute of Metals.....               | Boston, Mass.       | Sept. 24-29  |
| American Foundrymen's Association.....          | Boston, Mass.       | Sept. 24-29  |
| Chemical Industries, Third National Exhibition. | New York City       | Sept. 24-29  |
| American Electro-Chemical Society.....          | Pittsburgh, Pa.     | Oct. 3-6     |
| American Institute of Mining Engineers.....     | St. Louis, Mo.      | Oct. 8-13    |
| American Gas Institute.....                     | Washington, D. C.   | Oct. 16-19   |
| American Institute of Architects.....           | Philadelphia, Pa.   | Dec. 26-29   |
| American Society of Mechanical Engineers.....   | New York City       | Dec. 26-29   |
| American Mining Congress State Chapter.....     | Phoenix, Ariz.      | Dec. 26-29   |

## LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

HARRISON W. CRAVER, Director

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

## LIBRARY ACCESSIONS

### PARTIAL LIST CLASSIFIED BY SUBJECTS

#### Mining and Metallurgy

ASPHYXIATION FROM BLAST FURNACE GAS. Technical Paper No. 106, U. S. Bureau of Mines, Washington, 1916.

LIMITS OF COMPLETE INFLAMMABILITY OF MIXTURES OF MINE GASES AND OF INDUSTRIAL GASES WITH AIR. Technical Paper No. 150, U. S. Bureau of Mines, Washington, 1917.

MINE TAXATION IN THE UNITED STATES. By L. E. Young. (University of Illinois Studies in the Social Sciences. Dec., 1916, Vol. V, No. 4). Urbana, 1916. (Gift of University of Illinois.)

OXYGEN MINE RESCUE APPARATUS AND PHYSIOLOGICAL EFFECTS ON USERS. Technical Paper No. 82, U. S. Bureau of Mines, Washington, 1917.

#### Geology and Mineral Resources

COLORADO BUREAU OF MINES. Biennial Report. 14th, 1915-16. Denver, 1916.

DIVINING ROD, A HISTORY OF WATER WITCHING, WITH A BIBLIOGRAPHY. Water Supply Paper No. 416, U. S. Geological Survey, Washington, 1917.

GEOLOGY OF SOUTHERN OHIO, INCLUDING JACKSON AND LAWRENCE COUNTIES AND PARTS OF PIKE, SCIOTO AND GALLIA. Bull. No. 20, Geological Survey of Ohio, 4th Ser.

- PRODUCTION OF IRON AND STEEL IN CANADA. 1915. Publication No. 419, Canada. Department of Mines, Ottawa, 1916.
- PROFILE SURVEYS IN THE COLORADO RIVER BASIN IN WYOMING, UTAH, COLORADO AND NEW MEXICO. Water Supply Paper No. 396, U. S. Geological Survey, Washington, 1917.
- REPORT ON THE BUILDING AND ORNAMENTAL STONES OF CANADA. Vol. IV, Ottawa, 1916.
- SANDSTONE QUARRYING IN THE UNITED STATES. Bull. No. 124, U. S. Bureau of Mines, Washington, 1917.
- STORY OF THE GRAND CANYON OF ARIZONA. By N. H. Darton, Kansas City, 1917. (Gift of Fred Harvey.)
- U. S. GEOLOGIC ATLAS. Eureka Springs—Harrison Folio, Arkansas-Missouri. (Folio 202) Washington, 1916.
- \_\_\_\_\_. Tolchester Folio, Maryland. (Folio 204) Washington, 1917.
- U. S. Geological Survey. Annual Report of the Director, 37th. Washington, 1916.

### Chemistry and Testing

- ENGINEERING CHEMISTRY. Ed. 3. By H. J. Phillips. London, 1902.
- EXPERIMENTAL ENGINEERING AND MANUAL FOR TESTING. By R. C. Carpenter. Ed. 6. New York, John Wiley & Sons, 1907. (Gift of Publisher.)

### General

- AMERICAN UNIFORM BOILER LAW SOCIETY. Condensed report of the American Uniform Boiler Code Congress, Washington, Dec. 4-6, 1916. Erie, Pa. 1916. (Gift of Society.)
- CHEMICAL GERMAN. An introduction to the study of German chemical literature. By F. C. Phillips. Ed. 2. Easton, Pa., 1915.
- COST KEEPING AND CONSTRUCTION ACCOUNTING. By G. Ed. Ross. Salem, Ore., 1917. (Gift of Author.)
- ENGINEERING DIRECTORY, 1917. Pt. 1-2. Chicago, Crawford Publishing Co., 1917.
- GEOGRAPHIC TABLES AND FORMULAS. Ed. 4. Bull. No. 650, U. S. Geological Survey, Washington, 1916.
- METAL SPINNING. By F. D. Crawshaw. Chicago, 1909.
- MODERN METHODS IN CONCRETE CONSTRUCTION. Bull. A-5, National Lime Manufacturers Association Hydrate Lime Bureau. Pittsburgh, 1917. (Gift of Association.)
- SCIENTIFIC MANAGEMENT. A list of references in the New York Public Library. Compiled by Walter Vail Brown. New York, 1917. (Gift of New York Public Library.)
- STRAIGHT LINE ENGINEERING DIAGRAMS. By Manifold and Poole. San Francisco, n.d.
- THE BEGINNINGS OF PORCELAIN IN CHINA. By Berthold Laufer, with a technical report by H. W. Nichols. Publication No. 192, Field Museum of Natural History, Chicago, 1917. (Gift of H. W. Nichols.)
- WHAT A GOOD GEOLOGIST CAN DO IN WAR. Prepared by R. A. F. Penrose, Jr., for the Geological Committee of the National Research Council. Philadelphia, 1917. (Gift of Author.)
- YUKON TERRITORY; ITS HISTORY AND RESOURCES. Ottawa, 1916. (Gift of Chester A. Thomas.)

### Company Reports

- UTAH COPPER COMPANY. Annual Report 12th, 1916. New York, 1917. (Gift of Company.)
- WALLABOO AND MOONTA MINING AND SMELTING COMPANY. Reports and statements of accounts. 27th annual report. Adelaide, 1916. (Gift of H. L. Hancock.)

### Trade Catalogs

- AMERICAN BLOWER COMPANY. Detroit, Mich. Commerical Value of Washed Air.
- AQUABAR COMPANY. Concrete tile roofing. Descriptive catalog. Philadelphia, n.d.
- \_\_\_\_\_. Specifications No. 11, covering the use of No. 1 compound in waterproofing mass concrete for foundation walls, subways, etc.
- \_\_\_\_\_. No. 12, covering the use of No. 1 compound in waterproofing masonry

and concrete walls and floors with cement plaster coat for foundation walls, floors, etc.

- \_\_\_\_\_. No. 15, covering the use of No. 1 compound for waterproof cement stucco on brick and hollow tile walls. (Gift of Aquabar Company.)
- DEARBORN CHEMICAL COMPANY. Chicago, Ill. Incrustation, corrosion, foaming, and other effects of water used in steam making and methods of prevention.
- DOMESTIC ENGINE AND PUMP COMPANY. Shippensburg, Pa. Bull. No. 160. Domestic gas engine driven contractors' machinery.
- HOMESTEAD VALVE MANUFACTURING COMPANY, Homestead, Pa. Homestead Quarter Turn Valves.
- NATIONAL TUBE COMPANY. Pittsburgh, Pa. National Bull. No. 26, 1916.
- RODNEY HUNT MACHINE COMPANY. Orange, Mass. Catalog No. 30. Hunt Water Controlling Apparatus.
- RYERSON, Jos. T. & SON. Chicago, Ill. The Prevention of Slipping and Falling Accidents with Safety Treads.
- SANDUSKY CEMENT COMPANY. Cleveland, Ohio. Medusa Waterproofed White Portland Cement. March, 1917. Medusa White Portland Cement. Stainless, perfectly white in color. 48 pp.
- STANDARD PUMP AND ENGINE COMPANY. Akron, Ohio. Catalog No. 16. Standard Pumping Units. Water Supply.
- WESTINGHOUSE ELECTRIC AND MANUFACTURING CO. East Pittsburgh, Pa. Catalog No. 38. Small motors and generators. March, 1917.

### Book Notices

Unless otherwise specified, books in the following list have been donated by the publishers. The Institute does not assume responsibility for any statements made. They are taken either from the preface or the text of the book.

- THE WORLD'S MINERALS. By Leonard J. Spencer, with an appendix by W. D. H. Hamman. N. Y., Frederick A. Stokes Co., 1916. 11 + 327 pp., 8 × 6 in., 40 pl., 21 diagrams, cloth, \$2.75.

An attempt to present a popular and readable account, in the main confined to the 116 species of the more common simple minerals, which will help the student collector to identify his own specimens. Forty color-plates are included.

- MINE GASES AND VENTILATION. A Reference Handbook Combining Theory and Practice of Coal Mining. Designed to meet the needs of all students of coal mining, including mining engineers, mine superintendents, foremen, firebosses, shotfirers and miners preparing for examination for certificates of competency. By J. T. Beard. N. Y., Hill Publishing Co., 1916. 206 pp., 7 × 5 in., flexible cloth, \$2. (Gift of the author.)

Composed of material on the atmosphere, gases and ventilation of mines which has appeared in pocket book form, in the "Study Course in Coal Mining" department of "Coal Age" since March, 1913, and which was prepared in response to requests from coal miners who wished to know the development of formulas, the explanation of principles and the most approved and generally adopted methods. Contents: Air; Heat; Mine Gases; Theory of Ventilation; Practical Ventilation; Addenda.

- A HANDBOOK OF BRIQUETTING. By G. Franke. Translated by Fred C. A. H. Lantzberry, vol. 1. The Briquetting of Coals, Brown Coals, and Other Fuels. Phila., J. B. Lippincott Co.; Lond., Charles Griffin and Co., Ltd., 1917. 19 + 631 pp., 9 × 6 in., 225 illus., 9 pl., cloth, \$9.

This book is not only intended as a handbook for those engaged in the industry, but can be used as a text-book in schools of mining and metallurgy and technical high schools. The translator adds that the object of the translation is to put in the possession of English industrial leaders, knowledge which other countries have been compelled to acquire.

- FRYE'S TABLES FOR ASCERTAINING THE VALUE OF GOLD-QUARTZ SPECIMENS. By Jason S. Frye. Downieville, Cal., Jason S. Frye (privately printed) 1916. 56 pp., 4 × 3 in., leather, \$1. (Gift of the author.)

Vest-pocket book of tables for finding the percentage of gold and the value per ounce from the specific gravity of gold-quartz specimens containing different amounts of gold.

- TUBE MILLING. A Treatise on the Practical Application of the Tube Mill to Metallurgical Problems. By Algernon Del Mar. N. Y., McGraw-Hill Book Co., Inc.; Lond., Hill Publishing Co., Ltd., 1917. 10 + 159 pp., 6 × 9 in., 70 illus., 1 pl., cloth, \$2.



Covers the use of the conical and cylindrical tube mills for grinding ores, indicating in detail the best means of obtaining capacity at the least cost, and describing recent installations. It is, the author states, the only book entirely devoted to the subject. Contents: General Description; Amalgamating with the Tube; Grinding Ores with the Tube Mill for Flotation; Crushing Efficiencies; The Use of Wrought Iron and Alloy Steel; Appendix.

**THE VENTILATION HANDBOOK.** The Principles and Practice of Ventilation as Applied to Furnace Heating; Ducts, Flues and Dampers for Gravity Heating; Fans and Fan Work for Ventilation and Hot Blast Heating. By Charles L. Hubbard. N. Y., Sheet Metal Publishing Co., 1916. 218 pp., 8 × 6 in., 137 illus., cloth, \$2.

A series of questions, answers and descriptions, with illustrations, arranged from a series of articles prepared for "Sheet Metal." Care has been taken to keep all descriptions and mathematical work well within the understanding of the student and beginner.

**LABORATORY MANUAL OF BITUMINOUS MATERIALS.** For the Use of Students in Highway Engineering. By Prevost Hubbard, N. Y., John Wiley & Sons; Lond., Chapman & Hall, Ltd., 1916. 11 + 153 pp., 9 × 6 in., 38 illus., cloth, \$1.50.

Certain fundamentals not strictly a part of laboratory work have been included as a guide to students not well versed in the nomenclature, classification, and uses of bituminous materials.

**MICROSCOPIC EXAMINATION OF STEEL.** By Henry Fay. Wiley Engineering Series No. 3. N. Y., John Wiley & Sons; Lond., Chapman & Hall, Ltd., 1917. 18 pp., 9 × 6 in., 32 pl., cloth, \$1.25.

The material contained in this volume was originally issued by the Ordnance Department U. S. A. and was intended for the exclusive use of inspectors of ordnance material, but is now published for the use of others interested in the inspection of steel. It is meant only to present a mere outline of metallographic methods illustrating typical examples, but is not for use as a text-book. It is intended particularly for those who are in need of help in the interpretation of results. Over two-thirds of the volume consists of full-page plates.

**THE STORY OF BETHLEHEM STEEL.** By Arundel Cotter. N. Y., The Moody Magazine and Book Co., 1916. 65 pp., 7½ × 5 in., 8 pl., cloth, 75 cts.

Contents: Town Founded by Moravian Colonists; When Schwab Went Down to Bethlehem; Bethlehem and the War Stock Boom; Schwab's Theories.

**STRESSES IN WIRE-WRAPPED GUNS AND IN GUN CARRIAGES.** By Lieut. Col. Colden L'H. Ruggles. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1916. 11 + 259 pp., 9 × 6 in., cloth, \$3.

Explains a number of the important engineering principles underlying the design of wire-wrapped guns and of gun carriages. Originally prepared for the use of the cadets of the U. S. Military Academy. Contents: Elastic Strength of Wire-Wrapped Guns; Determination of the Forces Brought upon the Principal Parts of the 3-inch Field Carriage and a Disappearing Gun Carriage by the Discharge of the Gun; Stresses in Parts of Gun Carriages; Toothed Gearing; Counter Recoil Springs.

**GASOLINE AND HOW TO USE IT.** By G. A. Burrell, Bost., Oil Statistical Society, Inc., 1916. 281 pp., 6 × 4½ in., 1 illus., cloth, \$1.50.

The preface states that the aim of this book is to provide an intelligent understanding of the use of gasoline, to assist the motorist and the farmer, and to give the history of gasoline and petroleum.

**GAS CHEMISTS' HANDBOOK.** Compiled by Technical Committee, Sub-Committee on Chemical Tests, 1916, of the American Gas Institute. C. C. Tutweiler, Chairman, A. F. Kunberger, Editor, N. Y., American Gas Institute. 354 pp., 9 × 6 in. 70 illus., cloth, \$3.50.

The present handbook, a revision of the one compiled in 1914, presents methods for sampling and testing the materials used in gas manufacture. Contents: Raw Materials; Products of Gas Manufacture; Impurities in Gas; Tar Products; Miscellaneous and Tables.

**HANDBOOK OF CASINGHEAD GAS.** By Henry P. Westcott. Erie, Metric Metal Works, 1916. 9 + 274 pp., 5 × 8 in., illus., cloth, \$2.50.

Methods, statistics, etc., intended to supply information on the processes used for

extracting gasoline from natural gas. Based on visits to many existing plants and a study of their reports. Contents: General Physical Properties of Casinghead Gas Wells: Construction of Pipe Lines: Measuring Casinghead Gas: Gasoline Plant, Compression Method; Gasoline Plant, Absorption Method; Transportation of Gasoline; Miscellaneous.

**AZIMUTH.** By George L. Hosmer. 2d ed. rev., N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1916. 5 + 73 pp., 5 × 7 in., 6 illus., leather, \$1.

This handbook is intended to present in compact form certain approximate methods of determining the true bearing of a line, together with the necessary rules and tables arranged in a simple manner so that they will be useful to the practical surveyor. This edition contains a new method for finding the azimuth by an observation on the pole star at any hour angle when the local time is known. The tables of the sun's declination have been extended to 1919 and new star maps are given.

**INSTRUCTIONS TO LOCATING ENGINEERS AND FIELD PARTIES.** By F. Lavis. N. Y., McGraw-Hill Book Co., Inc. (c. 1916) 44 pp., 9 × 6 in., 9 diagrams, cloth, \$1. (Gift of the author.)

Reproduced in abridged form from "Railroad Location Surveys and Estimates." Originally prepared for the use of field parties working under his direction in the United States, afterward modified for use in South America and finally prepared in the present form for use in China. They are intended to secure uniform practice in making surveys and in the compilation of the results in the form of maps, estimates, etc.

**THE BAROMETRICAL DETERMINATION OF HEIGHTS.** A Practical Method of Barometrical Levelling and Hypsometry for Surveyors and Mountain Climbers.

By F. J. B. Cordeiro. 2d ed. rev. and enl. N. Y., Spon & Chamberlain; Lond., E. & F. N. Spon, Ltd., 1917. 26 pp., 7 × 4 in., cloth, 50 cts.

This little volume was an essay originally entered in the Hodgkins Prize Competition held under the auspices of the Smithsonian Institution, and was awarded honorable mention.

**VALUATION, DEPRECIATION AND THE RATE-BASE.** By Carl Ewald Grunsky and Carl Ewald Grunsky, Jr. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1917. 8 + 387 pp., 9 × 6 in., cloth, \$4. (Gift of the author and publisher.)

The author states that this book is the result of personal contact with the valuation problem. Special consideration has been given to a discussion of the non-agreement of the actual life of articles which have a limited period of usefulness with their probable or normal life. The tables included are not only intended for valuation engineers, but for any one having problems of finance and bonding to solve. Contents: Introduction and General Notes; Definitions; Fundamental Principles; Essentials of Value; Elements which Reduce Value; The Effect of the Non-Agreement between Actual and Probable Life upon the Determination of Depreciation; The Purpose of the Appraisal; The Fixing of Rates; Possible Procedures when the Rates for a Public Service are to be Fixed; Notes on the Determination of the Value of Real Estate in Eminent Domain Proceedings and for Rate Fixing Purposes; The Value of a Water-Right and of Reservoir and Watershed Lands; The Accounting System; The Valuation of Mines and Oil Properties by C. E. Grunsky, Jr., Tables.

**MODERN UNDERPINNING.** Development, Methods and Typical Examples. By Lazarus White and Edmund Astley Prentis, Jr. (Wiley Engineering Series No. 2) N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1917. 12 + 94 pp., 9 × 6 in., 48 illus., 1 pl., cloth, \$1.50.

Intended to exhibit, by means of photographs and drawings, the essential steps in underpinning as illustrated by the methods used in subway construction in New York City. Only enough text to supplement the illustrations included. Contents: General Aspects; Development of Underpinning and Methods; Shores, Needles, and Foundation Reinforcements; Specific Examples of Underpinning; Appendix.

**STEAM POWER.** By C. F. Hirshfield and T. C. Ulbricht. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1916. 8 + 420 pp., 8 × 5 in., 91 illus., cloth, \$2.

An attempt to collect in a comparatively small book such parts of the field of steam power as should be familiar to engineers whose work does not require that they be conversant with the more complicated thermodynamic principles considered in advanced treatments. Mathematical treatment of the subject has been eliminated to

the greatest possible extent. Intended for use as a text-book by students of civil engineering and in teaching power plant operators.

**STEAM PIPING.** Its Economical Design and Correct Layout. By A. Langstaff Johnston, Jr. N. Y., The Engineering Magazine Co., 1916. 6 + 62 pp., 5 × 8, 3 illus., 4 diagrams, cloth, \$2.

Consolidated and revised from a series of articles published in the "Engineering Magazine" in 1915. Analyzes the factors governing the flow of steam in pipes, and presents a group of curves for use in solving the problems of practical installation and determining the most economical size of pipe to select for any given conditions. Contents: How to Find the Right Pipe Sizes; Special Conditions Affecting Low-Pressure Systems; and Savings Obtainable from Exhaust Steam.

**STEAM TURBINES.** A Treatise Covering U. S. Naval Practice. By G. J. Myers. Annapolis, The U. S. Naval Institute, 1917. 7 + 246 pp., 12 × 8 in., 179 illus., 23 diagrams, 9 pl., cloth, \$4.50.

This book has been prepared to meet the needs of an elementary treatise on steam turbines for the use of midshipmen at the U. S. Naval Academy, and deals mainly with types found in the U. S. Naval Service.

**ELECTRIC TRACTION.** A Treatise on the Application of Electric Power to Tramways and Railways. By A. T. Dover, N. Y., and Lond., Whittaker & Co., 1917. 18 + 667 pp., 9 × 6 in., 518 illus., 5 pl., cloth.

Intended for engineers and advanced students. Representative examples of modern tramway and railway practice are included, but detailed accounts of electrification have been omitted and generating stations and transmission lines have not been considered. Contains a bibliography on electrification, and a number of worked examples have been included in the text. Contents: Mechanics of Train Movement; Motors; Control; Auxiliary Apparatus; Rolling Stock; Detailed Study of Train Movement; Track and Overhead Construction; Distributing Systems and Sub-stations.

**SPONS' ELECTRICAL POCKET-BOOK.** A Reference Book of General Electrical Information, Formulæ and Tables for Practical Engineers. By Walter H. Molesworth, N. Y., Spon & Chamberlain; Lond., E. & F. N. Spon, Ltd., 1916. 488 pp., 4 × 7 in. 325 illus., cloth \$2.

This book has been written for practical engineers, and it has been the aim to treat all subjects concisely and to avoid intricate mathematics. Full metric conversion tables have been inserted.

**HOW TO FIND FACTORY COSTS.** By C. Bertrand Thompson, Chic., N. Y., Lond., A. W. Shaw Co., 191 pp., 10 × 7 in., 51 charts, 1 diagram, 1 tab., cloth, \$3.

It is stated that this book is broad enough to apply to all kinds of industries, and is intended to be useful to the accountant as well as the factory head.

**PRACTICAL SHEET METAL DUCT CONSTRUCTION.** A Treatise in the Construction and Erection of Heating and Ventilating Ducts. By William Neubecker, N. Y., The Sheet Metal Publishing Co., 1916. 194 pp., 8 × 6 in., 217 illus., cloth, \$2.

The plan of the present work is to take up each operation, and by means of descriptions (usually illustrated) to show all operations incident to the construction and erection of heating and ventilating ducts.

**FRENCH MEASURE AND ENGLISH EQUIVALENTS.** By John Brook, N. Y., Spon & Chamberlain; Lond., E. & F. N. Spon., 1917. 7 × 80 pp. 4 × 3, 40 cts.

Vest-pocket size. Gives the English equivalents of the metric measures of length and weight and of the old French, Prussian, Austrian and Russian measures of length. A table for reducing the usual vulgar fractions of the inch to decimals is also given. The English equivalents are given in decimal fractions, correct to six places, and approximately in vulgar fractions as well. Intended for engineers, manufacturers and workmen.

**INDUSTRIAL PREPAREDNESS.** By C. E. Knoeppel, N. Y., The Engineering Magazine Co., 1916. 6 + 145 pp., 8 × 5 in., cloth, \$1.

A study of Germany's military and industrial preparedness intended to point the way to national greatness through the right kind of social, industrial and military preparedness.

## MEMBERSHIP

## NEW MEMBERS

The following list comprises the names of those persons who became members during the period of May 10, 1917, to June 10, 1917.

- ANDERSON, JAMES P., Mgr. of the Mines of the Ringwood Co.,  
50 Church St., New York, N. Y.
- BAKER, THOMAS C., Mine Supt., Cia. Santa Gertrudis, S. A.  
Pachuca, Hidalgo, Mexico.
- BARCUS, W. J. E., Min. Engr., Mining and Metallurgical Dept.,  
General Chemical Co., 25 Broad St., New York, N. Y.
- BENNETT, GEORGE N., Min. Engr., Butte & Superior Mining Co.,  
616 West Gold St., Butte, Mont.
- BROCKMANN, GUILLERMO, Banker, Gmo. Brockmann & Co., Room 3115,  
120 Broadway, New York, N. Y.
- BROWN, CLYDE L., Mining, Box 703, Goldfield, Nev.
- BRUNSKOG, CARL W., Draftsman, Consolidated Arizona Smelting Co.,  
Humboldt, Ariz.
- BURRELL, OSSIAN C., Mgr., American Zinc Co. of Tenn., Mascot, Tenn.
- CALKINS, FREDERICK E., Mining Geol. and Engr., General Mgr.,  
Arizona Exploration Corp., Miami, Ariz.
- CARTER, FREDERIC E., Physical Met., Baker Platinum Works,  
Austin & Murray Streets, Newark, N. J.
- CHRISTENSEN, H. L., Mill Supt., Alaska Juneau Gold Min. Co.,  
P. O. Box 1103, Juneau, Alaska.
- CLARK, R. ELLIOTT, Min. Engr., Bartlesville Zinc Co., 210 Seneca Ave.,  
Bartlesville, Okla.
- COLEMAN, FRANK J., Mine Foreman, Inspiration Cons. Copper Co., Miami, Ariz.
- COOPER, JAMES B., Sales Engr., Allis-Chalmers Manufacturing Co., Milwaukee, Wis.
- COOPER, LLOYD D., Chief Engr., E. J. Longyear Co., 710 Security Bldg.,  
Minneapolis, Minn.
- COWAN, WILLIAM W., Solution Man in Mill, McIntyre Porcupine Mines,  
Schumacher, Ont., Canada.
- CRUTCHER, ERNEST R., Box 651, Great Falls, Mont.
- CURRENS, WARREN W., Chemist, Ray Cons. Copper Co., Box 101, Ray, Ariz.
- DAVIS, CHARLES B., Engr., Machy. Salesman, C. B. Davis Engineering Co.,  
735 Brown Marx Bldg., Birmingham, Ala.
- DIXON, P. ST. JOHN, Min. & Met. The Dome Mines Co., Ltd., South Porcupine,  
Ontario, Canada.
- EASTON, WILLIAM B., Vice-Pres. and Chief Engr., Chalmers & Williams,  
1631 Edison Bldg., Chicago, Ill.
- EISENHAUER, RANDOLPH C., Min. Engr., Pilgrim Mining Co., Inc., Basin, Mont.
- ELLSWORTH, JOHN THAYER, Genl. Foreman, Elect. Zinc Plant, Anaconda Copper Min.  
Co., Boston and Montana Reduction Dept., Great Falls, Mont.
- ERWIN, ROBERT WESLEY, Genl. Supt., Mississippi Valley Iron Co., Waukon, Iowa.
- FORBES, JOHN J. V., Junior Min. Engr., U. S. Bureau of Mines,  
40th & Butler Streets, Pittsburgh, Pa.
- FOX, JOHN M., Min. Engr., Pine Terrace Inn, Dover, N. J.
- GARDNER, JAMES H., Petroleum Geol. and Appraising Engr.,  
510-11 New Daniel Bldg., Tulsa, Okla.
- GROLE, ARTHUR LAWRENCE, Representative in Mich.-Wis. district,  
E. J. Longyear Co., of Minneapolis, Minn., Crystal Falls, Mich.
- GILMAN, OSCAR W., Mine Foreman, Calumet & Arizona Copper Co., Lowell, Ariz.
- GROSVENOR, GEORGE L., Chief Engr., Great Western Smelters Corp., Mayor, Ariz.
- HARTLEY, BURTON, Geol., Cosden & Co., Box P., Tulsa, Okla.
- HIEBER, JOHN H., Min. & Concentrating Engr., Boston & Corbin Mining Co.,  
Corbin, Mont.
- HOWE, JAMES VANCE, Geol., Oklahoma Natural Gas Co., 211 World Bldg.,  
Tulsa, Okla.
- HOYT, J. C., Mining Engr., Box 102, Jerome, Ariz.
- HULL, CECIL B., Instructed to hold everything.

- IWAI, KYOSUKE, Genl. Supt., Yusenji Copper Mine, The Takeuchi Mining Co., Ltd.,  
Kokubumura, Nomigun, Ishikawaken, Japan.
- JOHNSON, EDWARD M., Supt., Eagle Picher Lead Co., Box 593, Henryetta, Okla.
- JOHNSTON, FRED L., Mech. Engr., The Western Chemical Mfg. Co., Denver, Colo.
- JOHNSTON, JOHN, Chemist, Research Dept., American Zinc, Lead & Smelt. Co.,  
3759 West Pine Blvd., St. Louis, Mo.
- JOHNSON, THOMAS B., Assayer, Mine and Stock Broker, Johnson Assay Co.,  
Box 570, El Paso, Tex.
- KATO, OHOKICHI, Min. Engr., Metal Mining Dept., Mitsu-Bishi Co., Tokyo, Japan.
- KENNEDY, HAROLD DE SAULLES, Mill Supt., Boston-American Mining Co.,  
Commerce Bldg., Everett, Wash.
- KIRCHEN, CHARLES.....Pres. Kirchen Mines Corp., Reno, Nevada.
- KISKADDON, W. W., Petroleum Geol., Kiskaddon Bros., Oil Producers,  
414-415 Daniels Bldg., Tulsa, Okla.
- KNOX, JOHN.....Genl. Supt., Calumet & Hecla Mining Co., Calumet, Mich.
- KRAUT, MAX, Min. and Met. Engr., Southwestern Engineering Co.,  
524 Wesley Roberts Bldg., Los Angeles, Cal.
- LEE, ARTHUR L., Engr., Homestead-Iron Dyke Mines Co., Inc., Box 86,  
Homestead, Ore.
- LEIGHTON, HENRY, Professor of Economic Geology, Univ. of Pittsburgh,  
Pittsburgh, Pa.
- LINDHOLM, MILTON S., Mining Engr., Phelps Dodge Corp., Copper Queen Branch,  
Box 2404, Bisbee, Ariz.
- McLAINE, WILLIAM L., Supt., General Petroleum Corp., Lost Hills, Kern Co., Cal.
- MANWARING, JOSEPH H., Supt., Butte-Ballaklava Copper Co., Box 6, Butte, Mont.
- MARQUARDT, ERNEST, Mining and Geological Engr., Associated Geological Engineers,  
120 Broadway, New York, N. Y.
- MONTEITH, ARCHIE LEON, Supt. Underground Limestone Quarry,  
The Low Moor Iron Co. of Va., Low Moor, Va.
- MUNN, M. J.....410 Clinton Bldg., Tulsa, Okla.
- NELSON, WILBUR A., Economic Geol., Secty. and Treas., Paga Mining Co.,  
Cartersville, Ga.
- NOWLAN, HARRY HACKETT, Geol., Valerius, McNutt and Hughes,  
329 Mayo Bldg., Tulsa, Okla.
- OHDAIRA, KOMATSUCHI, Mgr., Besshi Copper Mine, Niihama, Iyo Province, Japan.
- ORMSBEE, RANDALL H.....Mining Engr., Christmas, Ariz.
- PALMER, LEROY A., Mineral Examiner, U. S. General Land Office,  
512 Custom House, San Francisco, Cal.
- PALMER, ROBERT N., Supt. Worthington Mine, The Mond Nickel Co., Ltd.,  
Worthington, Ont., Canada.
- PALMER, WILLIAM R., Electrical and Mechanical Engr., 1801 East Vernon Ave.,  
Los Angeles, Cal.
- PARKER, D. J., Mine Safety Engr., U. S. Bureau of Mines,  
921 N. Euclid Ave., Pittsburgh, Pa.
- PATRON, ANTENOR RIZO, Min. and Met. Engr., Mgr., Negociacion Minera de Eulogio  
E. Fernandini, La Fundicion-Oficina Huarducaca, Peru, South America.
- PATTERSON, CLARENCE THOMAS.....Met., Latrobe Electric Steel Co., Latrobe, Pa.
- PIDGEON, GROVER C.....Mining Engr., Tejon Mining Co., Gleeson, Ariz.
- PITT, WILLIAM A., Genl. Mgr., Cauto Mining Co., San Nicolas, Santiago de Cuba,  
Cuba.
- PLUMMER, FREDERICK BYRON....Geol., Roxana Petroleum Co., Box L, Tulsa, Okla.
- PRINGLE, LEWIS B., Chief Chemist, Bonne Terre Division, St. Joseph Lead Co.,  
Bonne Terre, Mo.
- RAMAGE, JOHN H., Ensign, U. S. N. R. F., Metallurgist, Navy Yard, Norfolk, Va.
- REBER, LOUIS E., JR.....Geologist, United Verde Copper Co., Jerome, Ariz.
- RICHARDS, EDWIN R., Vice-Pres. and Mgr., Childers Leasing Co., Box 51,  
Midvale, Utah.
- RISHER, WILLIAM H., Supt., Huntington, Ark., District, Central Coal & Coke Co.,  
Box 22, Huntington, Ark.
- ROCHOW, W. F., Chemist, Harbison-Walker Refractories Co.,  
Farmers Bank Bldg., Pittsburgh, Pa.
- RODDER, JOHN CHRISTIAN.....Min. Engr., P. O. Box 994, Stockton, Cal.
- RYAN, WILLIAM E., Genl. Supt., The Vindicator Cons. Gold Mining Co.,  
Independence, Colo.
- SANDERSON, HAROLD H.....Seattle, Wash.
- SEONTS, SIDNEY L.....Wallace, Idaho.

- SNELL, LESTER W., Supt., Memphis Red River Mining Co., Red River, N. Mex.  
 ST. CLAIR, STUART, Cons. Geol. and Engr., Suite 800, 20 East Jackson Blvd.,  
 Chicago, Ill.
- STEE, CLARENCE O., Min. Engr., Foreman of the Esperanza Mines,  
 Cerro de Pasco Mining Co., Cerro de Pasco, Peru, South America.
- STENGL, RUDOLPH L., Supt. Century Zinc Co., 416 W. 17th St., Joplin, Mo.
- STONE, FRANK M., Master Mechanic & Chief Electrician,  
 Silver King Coalition Mines Co., Park City, Utah.
- STONE, GALEN L., Hayden, Stone & Co., 87 Milk St., Boston, Mass.
- SWAIN, SETH L., Met. Engr., P. O. Box 117, East Orange, N. J.
- TAGGART, OLIVER R., Chemist, St. Louis Smelting & Refining Co., Collinsville, Ill.
- TAPLIN, THOMAS J., JR., Spassky Zavod, Karagandy Post Office,  
 Akmolinsk District, Siberia.
- THOMPSON, A. PAUL, Asst. Chemist, Electrolytic Zinc Plant,  
 Great Falls Reduction Works A. C. M. Co., Great Falls, Mont.
- THOMPSON, J. W., Met. Engr., General Engineering Co., 159 Pierpont St.,  
 Salt Lake City, Utah.
- TRAPHAGEN, HENRY, Met. and Chemist, High Speed Tools Corp., 18 Broadway,  
 Toledo, Ohio.
- TUNNELL, H. R., Foreman, Pennsylvania Mine, Anaconda Copper Mining Co.,  
 Butte, Mont.
- VAZQUEZ, D. A., Asst. Engr., The Spanish American Iron Co., Felton, Oriente, Cuba.
- WASHBAUGH, WALTER, Genl. Supt. and Civil Engr., Gambio-Clinton Mining Corp.,  
 209 N. 21st Street, Birmingham, Ala.
- WATKINS, JOSEPH C., Mgr., McDonald Land & Mining Co.,  
 Rooms 430-431, Frisco Bldg., Joplin, Mo.
- WATROUS, MARK U., Min. Engr., Balaklala Cons. Copper Co.,  
 Coram, Shasta Co., Cal.
- WRIGHT, THOMAS W., Shift Boss, Calumet & Arizona Mining Co.,  
 Box 3754, Lowell, Ariz.

### Associate Members

- DODD, ALFRED W., American Zinc, Lead & Smelting Co., 120 Broadway,  
 New York, N. Y.
- HANNON, C. W., Salesman, Sullivan Machinery Co., 461 Market St.,  
 San Francisco, Cal.
- MARK, GEORGE W., Mgr., Missouri Mohave Mining & Milling Co.,  
 Cerbat, Mohave County, Ariz.
- SHAW, CHARLES H., Attorney at Law and Petroleum Engr.,  
 P. O. Box 206, Lawton, Okla.

### Junior Members

- CHASE, J. L., Geol., Kansas Natural Gas Co., Bartlesville, Okla.
- CONSTINE, JOHN, Student, Lehigh University, Price Hall, So. Bethlehem, Pa.
- DIXON, MAX M., Student, Columbia School of Mines, New York, N. Y.
- FELEDY, CHARLES FRANK, Met. Engr., Pennsylvania State College, State College, Pa.
- HEINE, BERNHARDT E., 126 Market St., Mt. Clemens, Mich.
- HOSFORD, HARRY W., Lieutenant U. S. Navy, Navy Dept., Washington, D. C.
- HUEH, KWEI LUN, Student, Massachusetts Institute of Technology,  
 Cambridge, Mass.
- HUBER, J. EARL, Student, Carnegie Institute of Technology, Pittsburgh, Pa.
- KLUGESCHIED, WALTER P., Min. Engr., Mine Efficiency Engr., Phelps-Dodge Co.,  
 Burro Mt. Branch, Tyrone, N. Mex.
- LORAIN, SINCLAIR H., Student, Michigan College of Mines, 233 College Ave.,  
 Houghton, Mich.
- MACMILLAN, HERBERT, Mining Engr., Columbia University, New York, N. Y.
- MEWHIRTER, SYDNEY A., Designate Lieut., Corps of Engineers, U. S. Reserve,  
 U. S. A., Ft. Leavenworth, Kansas.
- MIZEL, MAYNARD, Student, 302 Church St., Stevens Pt., Wis.
- PRICE, VIRGIL T., Student, So. Dakota School of Mines, Rapid City, So. Dak.
- WALSH, J. K., Student, 3507 Humphrey St., St. Louis, Mo.
- WINCHELL, JOHN H., JR., Engr., Deming Mines Co., Nampa, Idaho.

## CHANGE OF STATUS

*Associate to Member*

BOYLE, JOHN.....Status changed Associate Member to Member.

**Total Membership, June 10, 1917 .....6,366**

## CANDIDATES FOR MEMBERSHIP

**APPLICATION FOR MEMBERSHIP.**—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

*Applications Lacking Endorsement*

Applications for membership have been received from Messrs. Mackay and Trench, whose records are given below. These applications lack the necessary number of endorsers, but since these candidates live at some distance from the headquarters of the Institute, their records are published here in order that any members who are acquainted with them may be advised of the circumstances and may have an opportunity of writing to the Secretary endorsing these candidates.

**Alexander Dudley Mackay**, Waratah, Tasmania.

Proposed by J. D. Millen, Harold Rabling.

Born 1888, St. Lawrence, Queensland. 1903-06, Launceston Grammar School. 1907-09, University of Tasmania, B. Sc. 1910-12, University of Melbourne, B. M. E. 1916, Degree of M. M. E. 1912-13, Asst. to chief mining inspector, Mines Dept. of Victoria. 1913-16, Chief Assayer, engaged in research work, Mt. Bischoff Tin Mining Co.

Present position: Chief Assayer, Mt. Bischoff Tin Min. Co.

**Robert Hamilton Trench**, Rangoon, Burma, India.

Proposed by A. W. G. Bleack, John Cadman.

Born 1880, Liverpool. 1902, Univ. of Cambridge, England, 2d class Mechanical Science. 1902-04, British Westinghouse Electric & Mfg. Co., Trafford Park, Manchester. 1904-08, Asst. Mgr. and Mgr., London & Pacific Petroleum Co. 1908-12, general reporting and executive work on properties in Russia, West Indies, East Indies, Salina, United States, Messrs. Thompson and Hunter, London.

Present position—1912 to date: Genl. Mgr., British Burmah Petroleum Co.

The following persons have been proposed during the period May 10, 1917, to June 10, 1917, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

*Members*

**Alfred Alder**, Rapid City, So. Dak.

Proposed by A. W. Smith, J. Burns Read, Zay Jeffries.

Born 1888, Volin, So. Dak. 1913, Graduated South Dakota School of Mines. Fall 1913, Graduate Student So. Dak. School of Mines. 1913, Assayer, Otho Power & Development Co., Glendale, So. Dak. 1914-15, Mill Supt., Bismarck Cons. Mines Co., Deadwood, So. Dak. 1915-16, Millman, Refiner, Socorro M. & M. Co., Mogollon, N. Mex. 1916-17, Cyanide Foreman, Mill Supt., San Juan Mines Ltd., Mineral

del Salado, Rodeo, Prov. San Juan, Argentine, S. Amer.

Present position: Mill Supt., San Juan Mines Ltd.

**Robert Earl Baker**, Keokuk, Ia.

Proposed by Arthur Newell, de Courcy B. Browne, C. H. Taylor.

Born 1888, Columbus, Kans. 1908-12, University of Okla., Degree A. B., Major, Physics. 1912-14, Univ. of Ill., M. A., Chemistry; held an assistantship in Chemistry. 1914-15, Univ. of Wis., Ch. E.; held Fellowship in Chemical Engineering. 1915-16, Instructor in Chemical Engrg., Univ. of Cincinnati from Sept., 1915 to Mar., 1916. Mar. 1, 1916, Keokuk Electro-Metals Co., Keokuk, Iowa.

Present position: Works Mgr.

**Walter William Barnett**, Tul Mi Chung, Chosen, Korea.

Proposed by E. W. Mills, A. H. Collbran, J. F. Manning.

Born 1888, Cambridge, Ohio. 1902-06, West Denver High School. 1906-11, Colorado School of Mines; Degree, Engineer of Mines. 1911-15, Millman, Assayer and Surveyor for Wellington Mines Co., Breckenridge, Colo. 1915-16, Assayer and Surveyor for Ellamar Mining Co., Ellamar, Alaska. 1916, Seoul Mining Co.

Present position: Assayer and Chemist, Seoul Mining Co.

**Louis Arthur Barton**, Deerwood, Minn.

Proposed by J. Segall, J. E. Hodge, P. W. Donovan.

Born 1881, Detroit, Mich. 1887-95, Detroit Grade School. 1895-99, Detroit Central High School and Detroit University School. 1899-1903, University of Michigan, A. B. 1911, University of Wis., Mining Geology. 1903-04, Drillman with Cole & McDonald Exploration Co. 1904-05, Supt. in Hibbing District for Bradley Exploration Co. 1906-12, Genl. Supt., Rudolph Development Co. (Copper development for Weyerhaeuser & Co.) 1912-14, Exploration Eng. and Geologist for Cuyler Adams of Deerwood, Minn. 1914-17, Vice-Pres., Adbar Development Co., Deerwood, Minn.

Present position: Vice-Pres., Adbar Development Co.

**William S. Bates**, Butte, Mont.

Proposed by F. A. Linforth, M. H. Gidel, G. M. Fowler.

Born 1884, New York, N. Y. 1904, A. B., Columbia College. 1908, E. M., Columbia School of Mines. 1908-09, With American Surety Co., 100 Broadway, N. Y., as an Examiner of Fidelity Bond Applications. 1909-10, Partnership, Bozeman, Mont.; Locke & Dier, later Dier & Bates, Civil and Mining Engrs. 1910-12, With Sacket & Flaherty, Civil and Mining Engrs., Livingston, Mont. 1912 to date, with Anaconda Copper Mining Co., Butte, Mont.

Present position: Asst. Mining Engr., Anaconda Copper Mining Co.

**Adrien Berton**, Corocoro, Bolivia, South America.

Proposed by M. R. Lamb, W. R. Ingalls, R. H. Vail.

Born 1885, Versailles, France. 1905-08, Ecole Centrale des Arts et Manufactures (Paris).

Present position: Chief Engr., The Corocoro United Copper Mines Ltd.

**David Wells Boise**, Hurley, N. Mex.

Proposed by J. M. Sully, H. G. S. Anderson, O. Wiser.

Born 1883, Tower City, N. D. June, 1906, Graduated from North Dakota School of Mines with degree E. M. May, 1908, Completed a 6 months post graduate course at Colorado School of Mines. July, 1906, Employed by Santa Rita Mining Co., as Assayer and Surveyor. Worked in this capacity until Dec., 1907. July, 1908-Jan., 1909, Employed by Santa Rita Mining Co. at surveying, sampling, etc. Jan.-1909-June, 1909, Shift Boss in Test Mill, Santa Rita Min. Co. June, 1909-Sept., 1909, Shift Boss in Test Mill, Chino Copper Co. at Santa Rita, N. Mex. Oct., 1909-July, 1910, Clerk in mine office, Chino Copper Co., Santa Rita, N. Mex.

Present position—July, 1910, to date: Chief Clerk, Chino Copper Co.

**Allen Palmer Bradley**, Murray, Utah.

Proposed by W. W. Norton, C. W. Adams, Jr., A. L. Labbe.

Born 1892, Salt Lake City, Utah 1909-11, The Hill School, Pottstown, Pa. 1911-13, The Tome School, Port Deposit, Md. 1916, Sheffield Scientific School (Yale), Ph. B. 1917, 6 months post graduate work in metallurgy at Yale (Sheffield Scientific School). 1910, 3 months, Construction, Centennial Eureka Mine, Eureka, Utah. 1913, 2 months, Leonard-Baker Experimental Smoke Filter, Murray, Utah. (A. S. & R. Co.) 1916, 3 months, Smoke Chemist, Murray, Utah.

Present position: Smoke Chemist, American Smelting & Refining Co.



**Edmond E. Bronson**, New York, N. Y.

Proposed by H. Kohlberg, E. N. Skinner, C. F. Constant.

Born 1883, Oberlin, Ohio. 1903-06, Giroux Cons. Mines Co., Ely, Nev. Until 1911, Secretary of the Arizona-Nevada Copper Co. when it was converted into the Bagdad Copper Co. 1911 to date, Pres., Bagdad Copper Co., also interested in other exploration propositions in Luning, Nev., Darwin, Cal., and Sonora, Mexico.

Present position: Pres., Bagdad Copper Co.

**Niles Winfield Brook**, Dubuque, Ia.

Proposed by S. W. Burford, W. R. Smith, W. N. Smith.

Born 1888, Mankato, Minn. 1907-10, University of Wisconsin, General Engrg. 1908-09, Mining and Surveying-Drafting Inspector, Oliver Iron Min. Co., Coleraine, Minn. 1910-12, Draftsman, Inspector, Bookkeeper, Surveyor, Cleveland Min. Co., Hazel Green, Wis. 1913, Assayer, Cleveland Gold Min. Co., Dryden, Ont. 1913-14, Engr., Cleveland Min. Co., Hazel Green, Wis. 1915, Mgr., Saxe Pollard Min. Co., Linden, Wis.

Present position—1915 to date: Asst. Genl. Mgr., Lawrence Mines Co. and Cleveland Mining Co., Dubuque, Ia.

**James M. Burke**, Wakefield, Mass.

Proposed by C. H. Benedict, B. L. Miller, H. Eckfeldt.

Born 1890, Needham, Mass. 1910, Wakefield, Mass., High School. 1911, Worcester Academy. 1912, Powder Point School. 1916, Lehigh Univ., B. S. in Geology. 1916-17, Calumet & Hecla Mining Co., Efficiency work.

Present position: Efficiency work for Calumet & Hecla Min. Co.

**John Julius Cadot**, Salt Lake City, Utah.

Proposed by R. B. T. Kiliani, O. B. Hofstrand, W. J. Boudwin.

Born 1889, Salina, Kans. 1906-09, Howe Military School, Howe, Ind. 1909-15, Colorado School of Mines, E. M. degree. June to Sept. 1911, Homestake Mining Co., Lead, S. D. Summer 1912, Argo Reduction & Ore Purchasing Co., Idaho Springs, Colo. Summer 1913, Colo. National Guard, Engr. work, explosive testing, etc. March to Sept. 1914, Homestake Mining Co., Lead, S. D. 1915-17, Detroit Copper Co. & Arizona Copper Co., Morenci, Ariz.

Present position: The Hardinge Conical Mill Co.,

**Henry C. Carlisle**, New Rochelle, N. Y.

Proposed by J. E. Spurr, F. L. Oddie, E. N. Skinner, G. A. Carlisle, Jr.

Born 1886, East Orange, N. J. 1908, Mining Course, Sheffield Scientific School, Yale University, Degree Ph. B. 1910, School of Mines, Columbia University, Degree E. M. 1911-12, Nearly two years Ray Cons. Copper Co., Ray, Ariz. as surveyor, head sampler, stope eng., etc. 1912-13, In charge Stewart Copper property, Mina, Nevada, churn drillings under direction Mr. Sidney Ball. 1913, Underground work at mines in Tonopah, Nev. 1913-14, Examination work for six months for Tonopah Mining Co., in Bolivia, S. America. 1914-15, One year Nicaragua, C. A., on engineering staff J. E. Spurr for Tonopah Mining Co. 1916, On staff J. E. Spurr in northern Manitoba, Canada. 1917, Supt., Mandy Mining Co., a subsidiary of Tonopah Mining Co., operating copper mine north of The Pas, Manitoba.

Present position: Supt., Mandy Mining Co.

**Harry Charlton**, Taylorville, Ill.

Proposed by H. W. Nichols, F. W. De Wolf, E. C. Reeder.

Born 1874, Braidwood, Ill. High School, Wilmington, Ill. St. Viators College, Bourbonnais, Ill. Armour Institute. Never took Min. Engrg. in school or got a degree except as an Electrical Engr. My mining engrg. knowledge was gained in actual experience working with what were considered the best engineers in the business at that time. 1888-93, Worked with Jas. Anderson, M. E., of the Old Chicago, Wilmington & Vermillion Coal Co. and Oregon Improvement Co. Both now out of existence. Also with Herbert V. Williams of C. W. & V. Coal Co. 1894-1906, Did railroad work with the Wabash I. C. & Southern Pacific. 1905-06, Did some gold mine work in Cal. and Nev., mostly with the Charlton Cons. Min. Co., which I organized and projected. This property is still working. Went with the Pana Coal Co., Pana, Ill. latter part of 1906, as Supt., and did my own engineering.

Present position—1914 to date: Mining Engineer, Commonwealth Edison Co.

**Frederic Bagley Close**, Ann Arbor, Mich.

Proposed by A. B. Carpenter, C. C. Jones, F. A. Hale, Jr.

Born 1868, Hancock, Mich. 1886-89, University of Michigan. 1889-91, Michigan College of Mines, B. S. & E. M. 1891-93, Engineering practice at Hancock, Mich. 1893-94, Asst. Engr., Illinois Steel Co. 1894-99, Chief Engr., Chicago

Heights Land Assn., Chicago, Ill. 1899-1901, Chief Engr., Helvetia Copper Co., Helvetia, Ariz. 1901-02, Mine Supt. Helvetia Copper Co. 1902-07, Genl. Mgr., Helvetia Copper Co. 1907-08, Mine Supt. Pacific Smelting & Ref. Co. 1908-16, Cons. Engr. for Smith Interests, Detroit, Mich., and The White Pine Extension Copper Co.

Present position: Pres., The Tungsten Development Co.

**David Henderson Collier**, Kingman, Ariz.

Proposed by H. Kehoe, A. C. Overpeck, G. W. Kays.

Born 1878, Denver, Colo. 1886-94, Public School, Denver. 1894-98, High School, Denver. 1899-1901, Stanford University, Palo Alto, Cal. 1901-03, Chemist, Colorado Portland Cement Co., Portland, Colo. 1903-04, Assayer, Dos Cabezas Mining Co., Casas Grandes, Mex. 1904-12, Custom Assay Office, Guadalajara, Mex. 1913, Assayer, Needles plant, United States Smelting Co. 1913-14, Cyanide Supt., Frisco Gold Mines Co., Kingman, Ariz.

Present position: Supt., Thumb Butte Mines Co.

**Leonard S. Copelin**, Hayden, Ariz.

Proposed by L. S. Cates, D. D. Moffat, E. W. Engelmann.

Born 1888, Grand Harbor, N. Dak. 1907, Great Falls High School. 1907-08, Montana School of Mines. 1913, Missouri School of Mines, B. S. & E. M.; Anaconda Copper Co., B. & M. Reduction Plant, Great Falls, Mont. 1908-09, Sampler in Mills & Smelter. 1911-12, Investigator, A. C. M. Co., Butte, Mont. 1912-13, Asst. Instructor Chemistry, Missouri School of Mines. 1913-14, Mucker, Miner, Timberman, Tonopah Min. Co., Millers, Nev. 1914-15, Filter operator, Head Solution man, Experimental Laboratory.

Present position: Asst. Research Chemist, Ray Cons. Copper Co.

**Patrick Elmo Crane**, Greenwood, B. C.

Proposed by O. Lachmund, F. A. Thomson, H. E. Doelle, L. K. Armstrong.

Born 1883, O'Neill, Neb. 1906, Graduated Puget Sound Academy, Snohomish, Wash. 1911, B. S., Washington State College, Pullman, Wash. 1916, E. M., Washington State College. 1911, Assayer and Surveyor, West Coast Mining Co., Champion, Oregon. 1912-13, Mucker, Miner, Engr., Granby Cons. Mining & Smelt. Co., Phoenix, B. C. 1913-14, Engr., B. C. Copper Co., Greenwood, B. C. 1914-present Supt., Rawhide, Lone Star & Mother Lode Mines, Canada Copper Corp., Greenwood, B. C.

Present position: Supt., Rawhide, Lone Star & Mother Lode Mines, Canada Copper Corp.

**William Blake Crawford**, Bluefield, W. Va.

Proposed by J. J. Lincoln, H. D. Smith, R. A. Nowlin.

Born 1874, Sullivan Co., Tenn. High School education. 1889-94, Univ. of Tenn., Knoxville, Tenn. Civil Engrg. Special courses in mining and metallurgy, correspondence schools. 1894-96, Mining Engineering, Cripple Creek, Colo. 1897, R. R. preliminary and location work, Denver & Rio Grande R. R. 1897-98, Prospecting Walsenburg Coal Field. 1898-99, Lieut., 3rd U. S. V. Engineers, U. S. and Cuban service. 1900-04, Highway and Bridge Engr., Insular Government of Porto Rico. 1904-08, Coal operator, mines Black Raven Coal Co. and Crawford & Co., Four Mile, Bell Co., Ky. 1908-10, Cons. Engr., Knoxville, Tenn. 1910-13, Genl. Mgr., Va. Pocahontas (Carter Coal Co.) properties in W. Va. and Ky.

Present position—1913 to date: Cons. Engr.

**Thomas Davis**, Seattle, Wash.

Proposed by J. McD. Porter, G. B. Dennis, L. K. Armstrong.

Born 1855, Gloucester, Eng. 1868-1877, General Education in England. 1878-1917, General application to practical mining, Arizona, Idaho and British Columbia, specializing in coal.

Present position: Mining operations.

**Perry A. Delo**, Thane, Alaska.

Proposed by E. V. Daveler, J. Z. Bayless, G. T. Jackson.

Born 1886, Colorado Springs, Colo. 1899-1903, High School, Chicago. 1903-07, Chicago University 3½ years, B. A. and extra work in Chemistry. 1910-11, Extra work in Mathematics. 1908-09, C. M. & St. Paul Ry. Engrg. Dept. as Transitman, Levelman and Topographer. 1909-10, S. P. Ry Co., Resident Engineer. 1911-12, Miner and Prospector, Iditarod, Circle City, Alaska. 1912-13, Mining, Fairbanks and Dawson. 1913-14, Mining, Atlin, B. C. 1914, Miner and in Cyanide Plant, Treadwell Gold Mining Co. 1914-17, Oil Foreman and Fine Crushing Foreman, A. G. M. Co., Thane, Alaska.

Present position: Fine Crushing Foreman, Alaska Gastineau Gold Mining Co.

**Hohurs H. Dyer, Metcalf, Ariz.**

Proposed by R. B. Earling, J. D. Wanvig, Jr., J. E. Penberthy.

Born 1870, Wauseon, Ohio. 1887, Graduated High School, Ionia, Mich. 1891, Graduated Michigan College of Mines, Houghton, Mich. 1895, Degree Mining Engineer. 1891-95, Mining Engineer, Cleveland Cliffs Iron Co., Ishpeming, Mich. 1896-97, Mining Engr., Old Dominion Copper Co., Globe, Ariz. 1898-99, Mgr., Huson Iron Co., Desoto Iron Co., Crystal Falls, Mich. 1900, Mgr., J. Sellwood Interest, Binabik Mines. 1901-03, Experimental work, Park City Metals Co., Park City, Utah. 1903 to date, Supt. of Mines, Shannon Copper Co., Clifton, Ariz.

Present position: Mine Supt., Shannon Copper Co.

**Walter A. English, Washington, D. C.**

Proposed by David White, M. W. Ball, M. R. Campbell.

Born 1889, Los Angeles, Cal. 1907-10, University of California, B. S. 1911-13, University of California, M. S., Geol. August 1913 to present, U. S. Geological Survey work in California oilfields.

Present position: Asst. Geol., U. S. Geological Survey.

**David B. Gemmill, Crown King, Ariz.**

Proposed by E. S. Smith, J. N. D. Gray, D. H. Bradley, Jr.

Born 1879, Cayceus, Cal. 1897-99, Compton High School, Compton, Cal. 1904-05, Assayer and surveyor, Standard Mines, Cima, Cal. 1905-06, Supt., Bullfrog Red Oak Co., Rhyolite. 1907-09, Mgr., Boca del Cobre Mining Co., San Luis de Cordero, Drgo., Mexico. 1910-11, Genl. Mgr., West Mexican Mines, Ltd., Guadalupe y Calve, Chih., Mex. 1912-14, Pioche, Nevada and experimental flotation plant, Nev. Cons., Ely, Nev. 1915, Superior, Ariz. and Crow King, Ariz.

Present position: Mgr., Bradshaw Reduction Co.

**Julius W. Gerhard, St. Louis, Mo.**

Proposed by H. A. Wheeler, M. F. Chase, H. A. Buehler.

Born 1871, St. Louis, Mo. 1878, Public Schools, St. Louis. 1884, Smith Academy, St. Louis. 1890, Graduated Smith Academy. 1891, Bryant & Stratton Business College. I have read mining, metallurgical and chemical journals and books since leaving school. 1891, Cole & Glass Mfg. Co., Planing Mill, St. Louis. 1892, Granite Mineral Wool Co., St. Louis, Mo. 1898, 2nd Lieut, during Spanish-American War. 1899, "Kathleen R." Zinc Mine, Joplin, Mo. 1900, Magnetite Foundry Co., St. Louis, with Mr. F. Schwedtman and Mr. Frank Fogarty. 1902, Tramway Dept. of A. Leschen & Sons Rope Co. 1904, E. L. Adreon & Co., Railway Supplies, etc. 1907, Helped organize the Commercial Acid Co. and construct its plant and open up its mines.

Present position: Secretary, Commercial Acid Co.

**Ralph T. Godbe, Eureka, Utah.**

Proposed by W. Fitch, C. W. Stimpson, C. Fitch.

Born 1888, Salt Lake City, Utah. Public School, Salt Lake City, Utah. Surveyor for City of Salt Lake. Supt., Michigan Utah Mining Co. For number of years Asst. Mine Supt., Chief Consolidated Mining Co.

Present position: Asst. Supt., Chief Cons. Mining Co.

**Joseph C. Heilman, Wallace, Idaho.**

Proposed by I. C. Purington, D. F. Haley, R. J. White.

Born 1887, Montgomery, Pa. 1904-09, Lehigh University, Degree of E. M. 1909, Lehigh Coal & Navigation Co. as Transitman. 1910-11, Locating Engr., Washington State Highway Comm. 1912, Transitman, Utah Copper Co. 1913, Draftsman, Bunker Hill & Sullivan Min. & Concentrating Co. 1914-15-16, Min. Engr. and Mineral Surveyor, Merriam & Merriam Mining Engrs. at Wallace, Idaho. 1917, Mining Engr., American Metal Co., Ltd.

Present position: Min. Engr., representing the American Metal Co., Ltd.

**Joseph Samuel Henry, Castlemaine, Vic., Australia.**

Proposed by N. Dickerman, N. Cleaveland, W. C. Hammon.

Born 1876, Hill Plain. 1884-91, General in several of the Public Schools of the State of Victoria, Aust. 1891-96, Technical in the Technical College of Geelong, Vic., Aust. 1894-99 Messrs. Humble & Nicholson, Geelong, Vic., Aust. 1900-04, Messrs. Bucknall Bros., Leadenhall St., London, Eng. 1904-06, Messrs. Thompson & Co., Castlemaine, Vic., Aust. Manufacturers of Min. Machinery, etc. 1906-09, Chief Engr., Davis & Kershaw, Grahamstown, N. S. W., Aust., Suction pump dredging for gold. 1910-12, Genl. Mgr. and Engr., Tongkah Compound Tin Dredging Co., Puket, Siam, Office Collins St., Melbourne, Aust. 1912 to date, Cons. Engr., Kampong Kamunting Tin Dredging Co., Taiping, Perak, F. M. S., Office Challis House,

Sydney, Aust. 1915 to date, Cons. Engr. Asam Kumbang Tin Dredging Co., Taiping, Perak, F. M. S. The last companies are bucket dredging.

Present position: Consulting Engineer.

**Charles A. Herbert**, Pittsburgh, Pa.

Proposed by J. C. Roberts, F. Carroll, F. H. Bostwick.

Born 1879, Saginaw, Mich. 1901, Graduated Mich. College of Mines, Houghton, Mich. Degrees of B. S. and E. M. 1901-05, Min. Engr., Chicago Wilmington & Vermillion Coal Co., Chicago. 1905-15, Supt. and Genl. Supt., Chicago Wilmington & Vermillion Coal Co. 1915-17, Min. Engr., La Salle County Carbon Coal Co., La Salle, Wis.

Present position: Coal Mining Engr., U. S. Bureau of Mines.

**Guy Byron Hognason**, Hayden, Ariz.

Proposed by D. C. Jackling, D. D. Moffat, W. I. Garms.

Born 1886, Clarkfield, Minn. 1909, University of Minn., E. M. 1909-10, Engr., Oliver Iron Min. Co., Hibbing, Minn. 1910-11, Engr., Utah Copper Co., Bingham, Utah. 1911-12, Engr., O. W. N., Wash. 1912-17, Millman, Ray Cons. Copper Co., Hayden, Ariz.

Present position: Concentrator Foreman, Ray Cons. Copper Co.

**Curtis Clay Hubbard**, Gillespie, Ill.

Proposed by F. F. Jorgensen, H. H. Stoek, E. A. Holbrook.

Born 1887, Joslin, Ill. 1905, Champaign High, Champaign, Ill. 1909, B. S. degree in Civil Engineering, University of Ill., Urbana, Ill. 1915, B. S. degree in Mining Engineering, University of Ill., Urbana, Ill. 1907, Miner, Copper Queen Co., Bisbee, Ariz. 1909, Transitman, R. R. Constr., C. B. & Q. R. R. 1910-11, Engr., on Constr. Rock Island Plow Co., Rock Island, Ill. 1912-14, Independent Surveyor, Rock Island, Ill. 1915, Field Engr., Illinois State Highway Dept., Springfield, Ill. 1916, Draftsman and designer, Illinois State Highway Dept. 1917, Asst. Engr., Superior Coal Co., Gillespie, Ill.

Present position: Asst. Engr., Superior Coal Co.

**Roy Arthur Hunt**, Pittsburgh, Pa.

Proposed by T. Allderdice, Hugh Archbald, G. F. Kunz.

Born 1881, Nashua, New Hampshire. 1903, Yale University, A. B. Went to Aluminum Co. of America, Pittsburgh, Pa. 14 years with this company.

Present position: General Supt.

**L. W. Hutton**, Spokane, Wash.

Proposed by J. C. Haas, M. J. Sweeny, L. K. Armstrong.

Born 1860, Libertyville, Iowa. 1903, Grad. Vandemiller Engrg. College, San Francisco, Cal. 1875, Common Schools in Iowa. 1885, Locomotive Engineering. 1897, Idaho and elsewhere. 1895-1917, Mine operator, Hercules, Gertie, Sherman, Apex Mines.

Present position: Mine Operator.

**Eugene Law Ickes**, Westminster, London, S. W. I., Eng.

Proposed by Roderic Crandall, P. C. A. Stewart, E. De Golyer.

Born 1885, Deadwood, So. Dak. 1909, Graduated College of Mining, University of Cal. 1909-10, Lab. Instructor, Petrography and Mineralogy, Univ. of Cal. 1910, Junior Geologist, U. S. Geological Survey. 1911-12, Geologist, Mexican Eagle Oil Co., Tampico, Mex.

Present position—1912 to date: Geologist, S. Pearson & Son, Ltd.

**Floyd M. Jardine**, Thane, Alaska.

Proposed by E. V. Daveler, John Z. Bayless, G. T. Jackson.

Born 1884, Ouray, Colo. 1911, Graduated from Colorado College. 1911-15, Sub-Foreman, Metallurgical Engr., General Mill Foreman, Ray Cons. Copper Co., Hayden, Ariz. 1915 to date, General Mill Foreman, Alaska Gastineau Mining Co., Thane, Alaska.

Present position: General Mill Foreman, Alaska Gastineau Min. Co.

**Arthur G. Johnson**, Baxter Springs, Kan.

Proposed by S. H. Davis, H. Tackmann, O. M. Bilharz.

Born 1891, Iron Mt., Mich. 1911-13, Wisconsin State Mining School. 1913-15, Underground for Century Zinc Co. 1915-16, Geologist for Philip N. Moore. 1916-17, Geologist for Vinegar Hill Zinc Co.

Present position: Geologist.

**Fred Hall Kay**, Urbana, Ill.

Proposed by F. W. De Wolf, H. H. Stoek, E. A. Holbrook.

Born 1885, Camp Point, Ill. 1903-07, University of Chicago, S. B. 1907-09, Member U. S. Geological Survey. 1909-10, Sierra Cons. Mines Co., Mex. 1910-11, Southern Pacific Co., San Francisco, Cal. 1911 to present, Asst. State Geologist, Illinois.

Present position: Asst. State Geologist, Illinois.

**Arthur Lakes, Jr.**, Spokane, Wash.

Proposed by P. N. Moore, C. W. Goodale, L. K. Armstrong, W. H. Linney.

Born 1885, Golden, Colo. 1908, Colorado School of Mines. 1908-10, General Development Co., N. Y. 1910-11, Editor *Mining Science*, Denver. 1911-12, Mgr., Wire Patch Mines, Breckenridge. 1911-12, Engr., Colorado Gold Dredging Co. 1912, Mgr., Ymir-Wilcox Dev. Co., Ltd.

Present position—1916 to date: Member of firm of Larson & Lakes.

**William H. McCrum**, Kansas City, Mo.

Proposed by L. R. Budrow, J. W. Malcolmson, W. E. McCourt.

Born 1877, Kansas City, Mo. 1895, Central High School, Kansas City, Mo. Experience in oil business and manufacture of gasoline from gas. Engaged in production of crude oil and petroleum products.

Present position: Pres., Diamond Gasoline Co. and of the Osage Gasoline Co.; Secy. and Treas. Apax Gasoline Co., and of the Atlas Petroleum Co.

**Alexander McFarlane**, Tul Mi Chung, Chosen, Korea.

Proposed by A. H. Collbran, E. W. Mills, J. F. Manning.

Born 1870, Buckingham, Canada. 1904-05, Shift Boss, Le Roi Mining Co., Rossland, B. C., Canada. 1906, Shift Boss, Oriental Cons. Min. Co., Korea. 1907-1908, Mine Foreman, Korean Exploration Co., Korea. 1909-10, Examination work, with Mr. W. S. Holloway, Korea and Japan. 1911, Mine Foreman, Chosen Gold Mines Ltd., Korea.

Present position—1912 to date: Mine Foreman, Seoul Mining Co., Holkol and Tul Mi Chung, Korea.

**W. R. McIntosh**, McAlester, Okla.

Proposed by J. C. Reid, G. M. Brown, J. J. Rutledge.

Born 1884, Choctaw Co., Okla. 1898-1904, Jones Academy, near Hartshorne, Okla. 1904-06, Durant Presbyterian College, Durant, Okla. 1906-10, General employment as book-keeper, salesman, drug clerk and pharmacist for A. Silverman & Co., Genl. Merchandise, Miller-Nelson Co. Gen. Merchandise, Bank of Grant, and R. S. Coxad, Druggist, all of Grant, Okla. 1910-11, In drug business for myself in Ft. Towson, Okla. doing all prescription work and book-keeping.

Present position—1911 to date: Appointed Mining Trustee for Choctaw Nation under commission of Pres. Taft and Sam'l Adams, Acting Sec. of Interior.

**Hugh Stewart McKnight**, Tul Mi Chung, Whang Hai Prov., Korea.

Proposed by E. W. Mills, A. H. Collbran, J. F. Manning.

Born 1891, Dallas, Tex. 1905-09, St. Matthews School, Dallas, Tex. 1909-11, University of the South, Sewanee, Tenn. 1911-15, Colorado School of Mines, Golden, Colo., Degree of E. M. 1915, North Sonora Copper Co., Sonora, Mex. 1915-17, Seou Mining Co., Suan Concession, Korea.

Present position: Mine Surveyor and Sampler.

**Paul Moore McKorkle**, Gillespie, Ill.

Proposed by F. F. Jorgensen, H. N. Eavenson, H. W. Saunders.

Born 1893, Center, Ind. 1913, H. S. Surveying and Mapping diploma. 1914, Finished over half I. C. S. Coal Mining. 1915, Special Junior Mining Student, Kentucky State University, Lexington, Ky. 1916, Finished I. C. S. Civil Engineering. 1913-17, Home study. Mathematics and Engineering subjects in home study Dept., University of Wisconsin, at present a student in Univ. of Wis. 1913, Rodman, Instrumentman, Consolidation Coal Co., Jenkins, Ky. 1914, Asst. Engr., County Engr., Lake County, Ind. 1915, Draftsman, Crystal Block Coal & Coke Co., Welch, W. Va. 1915-16, Mining Engr., Superior Coal Co., Gillespie, Ill. 1916, Draftsman, designer, Portland Cement Association, Conway Bldg., Chicago, Ill.

Present position: Asst. to Chief Engr., Superior Coal Co.

**John Dickerson Martin**, Clothier, W. Va.

Proposed by C. E. Krebs, J. M. Clark, G. H. Caperton.

Born 1890, Salem, Va. 1907-09, Virginia Polytechnic Institute, Blacksburg, Va. 1909-10, Virginia Iron, Coal & Coke Co., Roanoke, Va., as a chainman and transitman. 1910-11, Chain and Transitman, Tredwater Coal & Coke Co., Vivian, N. Va. 1911-1914, Asst. and Chief Engr., U. S. Coal & Oil Co. 1914-16, Chief Engr. and Supt. Ethel Coal Co., Ethel, N. Va.

Present position: Member of firm of Ewing & Martin, Civil, Mining and Consulting Engineers.

**Harold Blair Menardi**, Battle Mt., Nev.

Proposed by R. A. Harder, J. A. Carpenter, E. M. Bordwell, G. J. Young.

Born 1886, Buffalo, Wyo. 1908, A. B., Stanford University. 1909, P. G. Work, University of Nevada. 1911, Metals Recovery Co. 1912, Maricopa Mines Co. 1914, Cortez Mining & Reduction Co. 1916, H. C. Christensen. 1917, Nicklas Mining Co., Breitung & Co., Ltd., 11 Pine St., N. Y.

Present position: Manager.

**William P. Michell**, Marion, Ky.

Proposed by A. H. Reed, J. B. White, W. E. McCourt.

Born—, Wadebridge. General education. Floridin Co., 9 years, Mines and Mills. 1889-92, Rep. of Colombia, Govt. of Colombia, South America. 1898, Prospecting in South America. 1905, Prospecting in South America.

Present position—1916 to date: Haffarr Mining Co.

**John J. Moisan**, Bagnio, Benguet, P. I.

Proposed by C. M. Eye, E. L. Oliver, H. C. Wilmot.

Born 1886, French Corral, Cal. 1892-99, Grammar School, Walnut Creek Contra Costa. 1899-1901, High School. 1901-04, Polytechnic School B, San Francisco. 1904-05, Polytechnic School E, San Francisco, Cal. 1905-06, Private tutelage from M. L. Crafter. 1906-08, Healds E. College, Degree of Met. 1908-09, Testing, sampling, cyaniding ores. 1909-10, North Star Min. Co. 1910-11, Richmond Chemical Co., Sampling and Testing. 1911-13, Colorado M. Co., Assayer, Shift Boss, Draftsman. 1913-14, Crescent Mills Reduction Co., Plumas Co. 1914, Keystone Min. Co., Masbate, Philippines, Constructor, Mill Foreman, Millwright. 1914-16, Benguet Consolidated Min. Co., Benguet, P. I., as Draftsman, Construction Mill Foreman.

Present position: Mill Foreman, Benguet Consolidated Mining Co.

**Chester Naramore**, Washington, D. C.

Proposed by A. F. Lucas, V. H. Manning, D. A. Lyon.

Born 1879, Winnemucca, Nev. 1903, Graduated Stanford Univ. Department of Geology and Mining, Degree A. B. 1903-05, California State Mining Bureau. Nov., 1905-Oct., 1909, U. S. Geological Survey, in charge of Denver Office. 1909-11, at San Francisco with Mr. Yale, U. S. Geological Survey. 1911-14, in the oil fields of California, as Geologist and Field Supt. at Coalinga in charge of Consolidated Oil Lands Co., Coalinga Oil Fields Co., Homestead Development Co., Coalinga Syndicate Oil Co. Jan., 1914-July, 1914, Supt., Montebello Oil Co., Oak Ridge Oil Co., Ventura Oil Lands Co. (All in Ventura County). 1914-15, Field Mgr., Calgary Petroleum Products Co., Calgary, Alberta. 1915-16, Deputy State Oil and Gas Supervisor, Taft, Cal. 1916-Apr., '17, Petroleum Technologist, U. S. Bureau of Mines.

Present position: Chief Petroleum Technologist of U. S. Bureau of Mines.

**Arthur Desborough Nash**, Nelson, B. C.

Proposed by F. H. Skeels, J. McD. Porter, L. K. Armstrong.

Born 1872, England. 1885-86, Oregon Agric. College. 1887, Geol., University of Cal. 1888-89, Testing Dept., S. P. R. R., Sacto., Cal. 1890, Electricity, Oregon Agric. College. 1896-1903, Exploration, British Yukon Placer Operations. 1904-08, Exploration, Nevada & Cal. Manhattan Dexter Co.'s., Nevada. 1909-10, Exploration Yurivari Dist., Venezuela, Sonora, Chihuahua, Jalisco, Mex. 1911, Northern Cal., Exploration. 1912-15, Exploration, Southeastern Alaska and in Western Alaska. 1916, Le Pas Dist., Manitoba, Canada.

Present position: Explorations, Kootenay Dist., B. C.

**Elmer Horace Noel**, Parsons, Kans.

Proposed by J. J. Rutledge, G. M. Brown, S. J. Tonkin.

Born 1881, Pittsburg, Kans. 1908, Kansas State University, B. S. 1908-09, Asst. to Chief Engr. for Board of Improvements at Ft. Smith, Ark. 1906-07, Asst. Mining Engr., Central Coal & Coke Co., Pittsburg, Kans. 1909, Min. Engr., for M. K. & T. Ry. Coal Dept. 1915, Supt. Mines, M. K. & T. Ry. Coal Dept. 1910, Min. Engr., Richmond Camden Coal Co. 1906-13, Min. Engr., Mayer Coal Co. 1913, Mining Engr., Coalgate Co.

Present position: Min. Engr., Supt. Mine Property at Mineral, Kans.

**Michael Th. Ortin**, Nijni-Novgorod, Russia.

Proposed by C. E. Locke, C. R. Hayward, E. E. Bugbee, H. O. Hofman.

Born 1880. Elabuga, Gov't. of Viatka. 1891-99, "Realschule" of Elabuga, Viatka Gov't. 1899-1907, Institute of Tech., Petrograd. Graduated as Engr. of

Tech. (First degree). 1907-08, Asst. Supt. at Katav-Ivanovsky Iron Works, Ufa Gov't. 1908-10, Asst. Chief Met. at Blagodatny Copper and Lead Smelter, Blagodatny Mines, Perm Gov't. 1910-11, Met. and Asst. Genl. Mgr., Blagodatny Copper and Lead Mines and Smelter. 1911-13 Instructor in Metallurgy in Polytechnic Institute, Warsaw. 1913-15, Commissioned by Ministry of Commerce and Industry to the United States to study the ore dressing and metallurgical practice; took a special course at Massachusetts Institute of Technology. 1915, Asst. Professor in Ore Dressing at Polytechnic Institute, Nijni-Novgorod.

Present position: Commissioned by Russian Gov't. as Artillery Inspector in United States.

**August Paulsen, Spokane, Wash.**

Proposed by J. C. Haas, C. H. Goodsell, L. K. Armstrong.

Born 1871, Denmark. Common School Education. 25 Years as operator in Coeur d'Alene Mining District.

Present position: Joint owner Hercules Mine and joint owner Consumers Co.

**John B. Pierson, Mahanoy City, Pa.**

Proposed by E. E. Kaercher, G. B. Hadesty, Charles Enzian.

Born 1882, Corbin, Va. 1904, B. S., in Civil Engineering. 1905, Degree of Civil Engineer, Virginia Polytechnic Institute, Blacksburg, Va. 1905-08, Transitman, U. S. Geological Survey. 1908-16, Transitman, Philadelphia & Reading Coal & Iron Co.

Present position—1916 to date: Division Engr., Mahanoy City Division, Philadelphia & Reading Coal & Iron Co.

**John C. Quade, Canton, Ill.**

Proposed by C. M. Young, H. H. Stoek, E. A. Holbrook.

Born 1872, Moline, Ill. 1891, High School, Moline, Ill. 1895, Univ. of Ill., B. S. & C. E. 1895-96, Employed by A. N. Talbot, Champaign Sewerage System. 1896-97, Employed by West Chicago Park Comm. Genl. Construction. 1897-1900, City Engr., Kewanee, Ill. 1900-04, Western Tube Co., Kewanee, Ill., Structural and Misc. Constr. 1904-17, Big Creek Colliery Co., Chicago, Ill. Have been employed for the past 13 years exclusively in the coal mining industry and handled all phases of this work. Am at present Chief Engr., having 8 mines in operation.

Present position: Chief Engr., Big Creek Colliery Co.

**Charles F. Richardson, Sturgis, Ky.**

Proposed by E. McAuliffe, H. A. Buehler, H. A. Wheeler.

Born 1863, Watterford, Vt. General education and practical experience. Thirty-two years railroad service in mechanical dept. Fifteen years of this time as genl. officer in charge of fuel inspection and supervision of fuel purchases, Baltimore & Ohio Railroad, Frisco and Rock Island Railroads. 7 years as Vice-Pres. and Genl. Mgr., West Ky. Coal Co., operating nine mines in Western Ky. field.

Present position: Vice-Pres. and Genl. Mgr., West Kentucky Coal Co.

**Boris Alexandrovitch Rouleff, Petrograd, Russia.**

Proposed by F. W. Draper, E. J. Carlyle, K. F. Altio.

Born 1879, Moscow. Royal Technical School of Moscow. Degree Mechanical Engineer. 1901-06, Asst. Supt., Iron & Brass Foundry, Kolomna Locomotive Works. 1906-09, Supt. of the above. 1909-15, Genl. Mgr., Verk Isetz Corp'n. 1915-16, Genl. Mgr., Verk Isetz Corp'n. 1915-16, Genl. Mgr., Demidoff Mining Estate, Nijni, Tagil.

Present position—Jan. 1, 1917 to date: Managing Director, Verk Isetz Corp'n.

**Charles Edward Schurch, Henryetta, Okla.**

Proposed by G. M. Brown, J. J. Rutledge, S. J. Tonkin.

Born 1879, Merchantville, N. J. 1898-1901, Technical Course, Wyoming Seminary, Kingston, Pa. 1901-03, U. S. Coast and Geodetic Survey on primary triangulation along 98th Meridian from Anthony, Kan. to Floresville, Tex. 1904, Transitman, Pennsylvania Railroad. 1905-11, With Jefferson and Clearfield Coal & Iron Co. as Construction Engr., and Supt., John Reed, G. M.

Present position—1911 to date: Mining Engr., Oklahoma Coal Co., Dewar, Okla. and general work in Henryetta Coal Field.

**W. R. Shanklin, Joplin, Mo.**

Proposed by E. T. Lednum, E. F. L'Engle, A. O. Ihlseng.

Born 1886, Balto. Co., Maryland. Educated in Baltimore County and City, Maryland. Public School and City College. 1907, Took up Mining as profession

with practical experience in Missouri, Kansas, Oklahoma, and Arkansas, principally the Joplin District, as Miner, Surveyor and Engineer. 1907, Started as Rodman in the employ of McKee & Cook, Mining & Civil Engrs., Joplin, Mo. Continued as Rodman, Timberman, and Transitman for 5 or 6 years. 1910, City Engr., Monett, Mo., paving streets and general city improvements. 1911, Employed Frisco R. R. Maintenance of Ways as Levelman and Transitman. 1912, Mining in Joplin District, also worked as Mining Engr. for Local Mining Companies. 1913, opened office as Mining Engr., and Mine Surveyor, practicing a general Engineering and Surveying business, also as mine Supt. for Picher Lead Co. in Okla. and Zinc Corp'n. of Mo. 1917, Entire attention to own business.

Present position: Mining Engineer.

**Harvey F. Shilling**, Clairton, Pa.

Proposed by J. S. Unger, A. N. Diehl, C. F. W. Rys.

Born 1884, Troy, Ohio. 1904-08, Ohio State University. April to Sept. 1907, Carnegie Steel Co., Bellaire Laboratory. May, 1909 to Jan. 1914, Carnegie Steel Co., Bellaire, Furnace Clerk. Jan., 1914, Same company as Asst. Blast Furnace Supt. Mar., 1916, Carnegie Steel Co., Clairton Works.

Present position: Metallurgist.

**Ethelbert Shores**, Salt Lake City, Utah.

Proposed by C. W. Stimpson, P. T. Boise, C. W. Whitley.

Born 1888, La Plata, Mo. Salt Lake City High School. Stanford University, Cal. Mining Engineering course, University of Utah. Chino Copper Co. (Several years with Utah Copper Co. as Met. Engr.) Associated with Mr. Wm. Janney at Chino and Mr. Frank G. Janney at Utah Copper Co.

Present position: Met. Engr., Stimpson Equipment Co., in charge Janney Flotation Dept.

**Ornan Bridges Smart**, Ajo, Ariz.

Proposed by Henry A. Tobelmann, H. S. Montgomery, B. H. Cody.

Born 1888, Carrollton, Ill. 1907, Completed preparatory schools. 1907-08, Student, Draftsman, San Francisco, Cal. 1908-11, Student of Min. Engrg., Leland Stanford Jr. University, Palo Alto, Cal. 1911-13, Practised mining and structural engrg. Associated with Horace A. Cook under the firm name of Cook & Smart, Phoenix, Ariz. 1913-16, Practised mining and structural engr., associated with R. M. Peabody, Phoenix, Ariz.

Present position—1916 to date: Construction Foreman, Sulphur Dioxide plant for New Cornelia Copper Co.

**Theodore Vassillevich Smirnov**, Ekaterinburg, Russia.

Proposed by F. W. Draper, C. J. Carlyle, D. F. Altio.

Born 1869, Kazan, Kazan University, Degree not given. Finished course of chemistry. 1893-98, Supt. Distillation dept., Kreeinobnikoff Bros. Candle Works. 1898-01, Chief Chemist, Kyshtim Mining Works, and Acting Supt. Copper Smelter. 1901-04, Supt., Cyanide Plant, Kochkar. 1904-06, In war with Japan. 1906-08, Supt., Cyanide Plant in Soimmovak Valley, Kyshtim Estate. 1908, Entered employ of Verk Isetz Corp'n., Supt., Pushmar Copper Smelter.

Present position: Supt., Copper Smelter, Pushmar Works, Verk Isetz Corp'n.

**Roy Starbird**, Berkeley, Cal.

Proposed by F. H. Probert, A. C. Lawson, E. A. Hersam.

Born 1894, San Francisco, Cal. May, 1917, B.S., College of Mining, University of California. 1912-13, Engr., placer prospecting, Brazil Exploration Co., Rio de Janeiro, Brazil. Summers of 1914-15-16, Underground work, North Star Mines Co., Grass Valley, Cal.; Rochester Mines Co., Rochester, Nev.; Nevada Consolidated Copper Co., Ely, Nev.

Present position: Engr. in employ of Alaska Gold Mines Co., Juneau, Alaska.

**Raymond Henry Summer**, Wilmington, Delaware.

Proposed by E. T. Lednum, G. B. Corless, P. B. Butler.

Born 1889, Georgetown, Colo. 1909, Graduate North Denver High School. 1909-14, University of Utah, School of Mines, Degree of B. S. in 1914. 1911, Jarbidge-Pavlak Gold M. & M. Co., W. P. Catlin, Jarbidge, Nev., as miner, cyanide millman. 1912, University of Utah, School of Mines, Utah Copper Co., W. L. Riser & G. Ensign. 1913, Summer, mill operator, vanner man, table man, classifier man. 1914, Engaged in general and mining insurance. Jan. to June, 1916, Operating tung-



sten lease in Boulder Co., Colo. 1916-17, Montezuma M. & M. Co. Flotation operator and in charge of operations at Silver Wave Mine, Montezuma, Colo. Jan., 1917 E. I. Du Pont de Nemours & Co., Wilmington, Delaware. Technical Div., underground blasting and work pertaining thereto.

Present position: Technical Representative, Du Pont Co.

**Herbert Hoover Wallower**, Webb City, Mo.

Proposed by F. C. Wallower, M. Van Sicken, C. T. Orr.

Born 1891, Harrisburg, Pa. 1910, Lawrenceville School. 1914, Princeton University. 1915, Cornell University. 1910, Disbrow Mining Co., Webb City, Mo. 1912, Lancaster Mining Co., Webb City, Mo. 1916, Cumberland Mining & Milling Co.

Present position: Supt., Cumberland Mines, Cumberland Min. & Milling Co.

**George Edward Webber, Jr.**, Ross, Marion Co., Cal.

Proposed by G. E. Webber, G. P. Bartholomew, F. Hellmann.

Born 1885, Central City, So. Dak. 1899-1905, Belmont Academy, Belmont, Cal. 1905-09, Univ. of Cal., College of Mining. 1909-12, Sampling Dept., Shift Boss, Cyanide & Smelting Dept., Survey Dept., Crown Mines, Ltd., Johannesburg, So. Africa. 1913-15, Min. Engr., Assayer, etc., at Jualin, Alaska Mines Co., Jualin, Alaska. Assay & Survey Dept. at Butters Salvador Mines Co., Ltd., Salvador, C. America. 1915-17, Shift Boss, Safety Engr., and Asst. Mine Supt. at The Dome Mines Co., Ltd., So. Porcupine, Ont., Can.

Present position: Not engaged.

**Howard K. Welch**, New York, N. Y.

Proposed by W. W. Mein, John Stevenson, Jr., H. E. Crawford.

Born 1882, Butte, Mont. 1897-1901, St. Pauls School, Concord, N. H. 1901-04, Sheffield Scientific School of Yale University, Degree of Ph. B. 1901-06, Four years in employ of Anaconda Copper Mining Co. 1913-15, Two years with Alaska Consolidated Co. of Juneau, Alaska. 1915-17, Two years Carlisle Mining Co. of Steeple-rock, New Mex. Have done considerable exploration work for various companies and individuals in Alaska, Montana, Idaho, Nevada, Arizona, New Mexico, Utah, etc.

Present position: Genl. Mgr., Carlisle Mining Co.

**Clarence E. Wheelock**, Mogollon, N. Mex.

Proposed by A. W. Smith, J. Burns Read, Zay Jeffries.

Born 1891, Keystone, S. Dak. 1909, Graduated from Preparatory Department, South Dakota School of Mines. 1909-11, South Dakota School of Mines. 1912-13, South Dakota School of Mines. 1911-12, Socono Mining & Mill. Co. 1913-14, Engineering Dept., Socono Mining & Milling Co. 1914-15, Assayer, Bismark Cons. Mines Co., Deadwood, S. Dak. 1915-16, Mine Foreman, Deep Down Mine, Socono Mining & Milling Co., Mogollon, N. Mex. 1916 to date, Battery Foreman, Mine Supt., San Juan Mines, Ltd.

Present position: Mine Supt., San Juan Mines Ltd., Mineral del Salado, Rodeo, Prov. San Juan, Argentine, S. Amer.

**William Henry Wright**, Golden, Colo.

Proposed by Harry Wolf, J. C. Roberts, C. A. Chase.

Born 1879, Council Grove, Kansas. 1898, Graduated Aspen High School, Aspen, Colo. 1898-99, B. S. course Colorado College. 1899-1903, Colorado School of Mines; did not finish. 1916, Six months in Dept. of Research and Testing at C. S. M. as an Assistant. 1893, Sample boy, Durant Mine, Aspen, Colo. 1893-99, Powder monkey, nipper, mucker, miner, contractor in mines at Aspen, Colo., during vacations. 1899-1903, Machine man, contractor aerial tram man, ore sorter, specimen-boss on Smuggler-Union, Telluride, Colo. 1903-06, Assayer and Chemist and acted at times as Mine Sampler & Surveyor, Melter, and Asst. in mill tests, Tomboy Gold Mines Co., Telluride, Colo. 1906-07, Assayer & Chemist, Smuggler-Union Min. Co., Telluride, Colo. 1907-14, Supt., Ashburton Mining Co., Marigold, Cal. Since that time to date have been engaged in examination work for various parties, principally Mr. Wells.

Present position: Member of firm of Malm-Wolf Co., Mining & Metallurgical Engrs., First National Bank Bldg., Denver, Colo.

### *Associate Members*

**A. W. Crawford**, Carlinville, Ill.

Proposed by J. D. Robertson, F. F. Jorgensen, W. E. McCourt.

Born 1861, Godfrey, Ill. Common School. 1900-03, Supt., coal mines. 1907-08

Coal mine operator. 1917, Interest in copper mining. 1917, Interest in oil lands and oil wells. 1917, Interest in Coal Lands.

Present position: Operator Coal Lands and mining.

**R. T. Price, Muskogee, Okla.**

Proposed by J. J. Rutledge, H. M. Brown, S. J. Tonkin.

Born 1870, Hyde Park, Pa. Only common school education. Have been engaged in the mining business practically all my life. Have been employed as General Manager of various coal companies in Ohio for the past twenty years, namely, Eastern Ohio Coal Co., Cleveland, Ohio; Kennon Coal & Mining Co., Cleveland, Ohio; R. T. Price Coal Co., Cleveland, Ohio; Ohio & Pa. Coal Co., Cleveland, Ohio; Rice Coal Co., Cleveland, Ohio.

Present position: Have been located in Muskogee for the past two and half years as Vice Pres. and Genl. Mgr., of the Oklahoma Consolidated Coal Co., Consolidated Fuel Co. and American Smokeless Coal Co. and Pres. of the Crescent Coal Co.

**James Randolph Walker, Salt Lake City, Utah.**

Proposed by C. W. Stimpson, O. J. Salisbury, J. A. McCaskell.

Born 1879, Berkeley Springs, W. Va. Berkeley School, N. Y. City. Georgetown University. 1898-1901, Treasurer and Managing Editor of Cosmopolitan Pub. House. 1901-03, Treas. and Business Mgr., Locomobile Co. of America. 1903-08, Managing Editor, Cosmopolitan Magazine for R. Hearst. 1905-06, Editor, Times Magazine and Treas. and Editor of Twentieth Century House. 1906-17, Investments.

Present position: Gen. Mgr., United Filters Corp.

### *Junior Members*

**Charles Warren Campbell, Champaign, Ill.**

Proposed by H. H. Stoek, C. M. Young, E. A. Holbrook.

Born 1894, Coal City, Ill. Sept. 1913 to present, University of Illinois, Mining Engineering. July, 1911 to Aug. 1913, General Utility, office and mine of Wilmington Star Mining Co.

Present position: Student, University of Illinois.

**Frank D. Cannon, Butte, Mont.**

Proposed by C. H. Clapp, C. H. Bowman, Bancroft Gore.

Born 1890, Gibbonsville, Ida. 1898-1906, Gibbonsville Public School. 1909-13, Academy of Idaho, Pocatello, Idaho. 1913-17, Montana State School of Mines, Butte, Mont. 1909-13, Worked three months during 1909 as carman for Kitty Burton Gold Mining Co. at Neysses, Ida. Worked during vacation of years 1910-11-12 for same company as miner. 1913-17, Worked three nights a week and summer vacations as a miner for the Anaconda Copper Mining Co., Butte, Mont.

Present position: Student, Montana State School of Mines.

**Stanley B. Carlisle, Butte, Mont.**

Proposed by C. H. Clapp, C. H. Bowman, Bancroft Gore.

Born 1890, Nashville, Kans. 1905-10, Public School, Nashville, Kans. 1910, South Western Academy, Winfield, Kans. 1911-12, South Western College, Winfield, Kans. 1910-11-12-13, Teaching in Public School. 1913-17, School of Mines, Butte, Mont. Vacations of 1911 and 1912 spent buying and shipping horses from western to eastern states. 1913-17, Worked three nights a week and during vacation in mines of the Anaconda Copper Co., Butte, Mont.

Present position: Student, Montana State School of Mines.

**Chester Greenwood, Vallejo, Cal.**

Proposed by F. C. Lincoln, W. S. Palmer, J. C. Jones.

Born 1896, Vallejo, Cal. 1914, Graduated from Vallejo High School. Entered Jan. 1915, University of Nevada. Transferred to Michigan College of Mines Sept. 1916 and remained till Jan. 1917, when I returned to University of Nevada. Jan. 1914-Nov. 1914, with York Dredging Co., York, Alaska. June 1916-Nov. 1916, Lost River Tin Co., Lost River, Alaska.

Present position: Leaving for South America June 1st for English Co.

**Wells K. Gregg, Madison, Wis.**

Proposed by R. S. McCaffery, E. A. Barnard, W. O. Hotchkiss.

Born 1892, Milwaukee. University of Wisconsin, B. S. Summer Session, 1917. 1913-14, Chemist, Standard Oil Co., Pittsburgh, Pa.

Present position: Student, University of Wisconsin.

**Alfred Alphonse Holland**, Grand Rapids, Mich.

Proposed by W. E. Hopper, F. W. McNair, A. J. Houle.

Born 1894, Grand Rapids, Mich. 1912, C. C. N. S., Grand Rapids, Mich. 1913, Loyola Univ., Chicago, Ill. 1914-16, Michigan College of Mines. 1913, Surveying, Grand Rapids & Indiana Ry. Co. 1915, T. O. Williams, County Surveyor, Kent Co., Mich. 1915-16, Grand Rapids & Indiana Ry. Co.

Present position: Student, Michigan College of Mines.

**T. Ling**, Butte, Mont.

Proposed by C. H. Clapp, C. H. Bowman, Bancroft Gore.

Born 1888, Honan, China. 1907-12, Nanka High School, Tientsin, China. 1912-13, Ching-Hua College, Peking, China. 1913-14, Michigan College of Mines, Houghton, Mich. 1914-17, Montana State School of Mines, Butte, Mont. 1916-17, Miner, Poulin Mine, Anaconda Copper Mining Co.

Present position: Student.

**Raymond L. McCann**, Bethlehem, Pa.

Proposed by H. Eckfeldt, J. W. Richards, B. L. Miller.

Born 1896, Harrisburg, Pa. 1913, Harrisburg Technical H. S. Summer 1913, Constr. Dept., Pennsylvania Steel Co., Steelton, Pa. Summer 1915, Munitions Dept., Harrisburg Pipe & Pipe Bending Co., Harrisburg, Pa. Summer 1916, Bethlehem Steel Co., Evaluation Survey, Lebanon Plant, Lebanon, Pa.

Present position: Student, Lehigh University, So. Bethlehem, Pa.

**Maurice Johnson Reed**, Champaign, Ill.

Proposed by H. H. Stoek, E. A. Holbrook, C. W. Young.

Born 1897, Emerson, Ill. Sept. 1913 to June 1917, as Mining student University of Ill., June and July 1916, The Vinegar Hill Zinc Co., Office, Platteville, Wis.

Present position: Student, University of Ill.

**Thomas Harold Wilkinson**, Butte, Mont.

Proposed by C. H. Clapp, C. H. Bowman, Bancroft Gore.

Born 1893, Brandon, Manitoba, Canada. 1907-11, Butte High School. 1913-17, Montana State School of Mines. 1911-12, Miner, Tramway Mine, Anaconda Copper Mining Co. 1913-17, with same company.

Present position: Student, Senior, Montana State School of Mines.

## CHANGE OF STATUS

### *Junior to Member*

**Albert S. Konselman**, Globe, Ariz.

Proposed by B. L. Miller, H. S. Drinker, M. J. Elsing, R. W. Hart.

Born 1894, New York, N. Y. 1910, Public Schools, N. Y. City. 1912, Cooper Institute, New York City. 1915, Lehigh University. 1913, Lehigh Valley Coal Co. 1914, Geologist for Dr. B. L. Miller. 1915-17, Engr., Cananea Cons. Copper Co.

Present position—1917 to date: Supt., Miami Mother Lode Mining Co.

**Holman Thompson Marshall**, Ruth, Nevada.

Proposed by W. S. Larsh, M. Mansfield, F. C. Riser, Jr.

Born 1891, Sioux City, Iowa. 1905-09, High School, Sioux City, Iowa. 1909-14, Missouri School of Mines at Rolla, Mo. Degree of B. S. in M. E. 1913, July and Aug., Fireman at Steam Shovel Mine of Utah Copper Co. at Bingham. 1914-16, Various occupations at Magna Mill of the Utah Copper Co. including operation of the Janney Flotation Machine. 1916-17, Engr., Dutchman and Pacific Mines in American Fork Canyon, Utah.

Present position—1917 to date: On Engineering Staff of the Nevada Cons. Mining Co.

**Herbert Nelson Witt**, Goldfield, Nev.

Proposed by J. F. Brown 2nd., J. W. Hutchinson, E. A. Julian.

Born 1889, Boston, Mass. 1908-10, At Harvard Univ. 1910-13, Three years leave of absence from college in mines of U. S. Coast Geodetic Survey. 1913-15, At Harvard Univ. Granted S. B. Degree in June, 1915. 1915-16, Asst. in Geol. at Harvard. 1914-15, Graduate student and Asst. in Dept. of Geol. at University of Cal. 1910-13, Surveying Officer, U. S. Coast & Geodetic Survey in the Philippines

and in Alaska. 1913, Geol., Spring Valley Oil Field, Wyo. 1914, Summer at Copper Queen Mine, Bisbee, Ariz. 1915, Geol., Aurora Cons. Mines Co., Aurora, Nev. 1916, Geol., Goldfield Cons. Mines Co., Goldfield, Nev.  
Present position: Geol., Goldfield Cons. Mines Co.

### CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period May 10, 1917 to June 10, 1917.

This list together with the list published in Bulletin Nos. 121 to 126, January to June, 1917, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1917 and brings it up to the date of June 10, 1917.

ALSDORF, FREDERICK C. .... Hotel Santa Rita, Tucson, Ariz.  
AMBLER, HARRY A. .... 4050 Russell Ave., St. Louis, Mo.  
AUSTIN, JOHN W. .... Assayer, Utah Metal & Tunnel Co., Bingham, Utah.  
AVERY, JOHN, Mgr., Electrolytic Zinc Co. of Aust., Pty. Ltd., Hobart, Tasmania.  
BALLENBERG, ADOLF G., Morris Paper Mills, 1517 Conway Bldg.,  
111 W. Washington St., Chicago, Ill.  
BANKS, HUBERT R. .... Cons. Arizona Smelting Co., Humboldt, Ariz.  
BARNARD, ENOCH A. .... 612 West Fifth St., Anaconda, Mont.  
BARTON, JOS. .... Mascot, Tenn.  
BENSON, ALEXANDER J. .... Grosseto, Italy.  
BERTHIER, ULYSSES H., Cia. Minera Paloma y Cabrillas S. A., Higuera, Coah., Mex.  
BLOCK, JAMES A. .... Jerome, Ariz.  
BOWMAN, JOSEPH V., Chief Chemist, American Smelting & Refining Co.,  
Chanarcitos, Carrizal, Chile, South America.  
BOYER, SAMUEL L. .... Instructed to hold all mail.  
BRADLEY, GEORGE O., Mech. Engr., Bradley, Bruff & Labarthe,  
85 Second St., San Francisco, Cal.  
BRADLEY, RICHARD J. H. .... 160 Broadway, New York, N. Y.  
BRINSMADE, ROBERT B. .... 9a Galena I, Puebla, Pue., Mexico.  
BRYDEN, CHARLES L. .... 90 West St., New York, N. Y.  
BURDICK, CHARLES A. .... 15 Broad St., New York, N. Y.  
CAMPBELL, DONALD G., Min. Engr., Campbell Wells & Elmendorf,  
114 James St., Seattle, Wash.  
CARPENTER, EDWIN E. .... Supt., Nevada Wonder Mining Co., Wonder, Nev.  
CARPENTER, JAMES R. .... P. O. Box 420, Altoona, Pa.  
CHURCH, JOHN A., JR. .... 154 11th St., Long Island City, N. Y.  
COE, HARRISON S. .... The Dorr Co., 17 Battery Place, New York, N. Y.  
COLES, HENRY O., Flotation man, Inspiration Cons. Copper Co., Box 608,  
Miami, Ariz.  
COLLBRAN, ARTHUR H. .... Seoul Mining Co., Holkol, Whang Hai Province, Korea.  
COOPER, MAURICE D., Min. Engr. .... Brownsville, Pa.  
CORLESS, GEORGE B. .... Room 316 Frisco Bldg., Joplin, Mo.  
CROCKER, A. L. .... 601 North West Bank Bldg., Minneapolis, Minn.  
CROGIN, RODNEY S. .... Ray, Ariz.  
CROOK, WELTON J. .... Box 985, Stanford University, Cal.  
DAVIES, ROBERT G. .... 1101 Franklin St., Boise, Idaho.  
DAVIS, ROY H. .... 206 Elvian Ave., Pittsburgh, Pa., E. E.  
DE HORA, MANUEL H., Consulting Engr., 242 Lexington Ave., New York, N. Y.  
DICKINSON, ALBERT W., Chief Engr., Western Coal & Mining Co.,  
1161 Railway Exchange Bldg., St. Louis, Mo.  
DOBSON, CHRISTOPHER G. .... Ray, Ariz.  
DOLMAN, PHILLIP B. .... Min. Engr., Gypsie Oil Co., Tulsa, Okla.  
DUNSFORD, ENSOR R. .... Clothier, W. Va.  
DWYER, C. EUSTACE. .... Creede Exploration Co., Creede, Colo.  
EATON, EDWIN R. .... Marion, Va.  
EATON, WALTER J. .... Cia. Minera de Penoles S. A., Ojuela, Mexico.  
EMMONS, N. H. 2d .... Irwinton, Ga.  
ESTABROOK, EDWARD L., Geologist, 306 State Hall, Grant Blvd. & O'Hara St.,  
Pittsburgh, Pa.

- EUSTICE, RICHARD.....46 Austral Terrace, Malvern Adelaide, South Australia.  
 EYOB, DJEVAD.....422 W. 115th St., New York, N. Y.  
 FARNHAM, C. MASON, Cerro de Pasco Mining Co., Cerro de Pasco, Peru,  
 South America.  
 FERRARIS, ERMINIO, Min. Engr., Mgr., Società di Montoponi, Turin, Italy,  
 via XX Settembre 38 bis.  
 FREUDENBERG, WALTER H.....Metals Milling Co., Tar River, Okla.  
 FREYN, HEINRICH J.....1122 E. 52d St., Chicago, Ill.  
 FRIELINGHAUS, GEORGE.....Krupp Wks., Essen Ruhr, Germany  
 GARNETT, THOMAS H.....Supt. Kelly Mines, Empire Zinc Co., Kelly, N. Mex.  
 GOETZ, MOWRY E.....Box 984, Woodlawn, Pa.  
 GOODRICH, HAROLD B.....Tulsa, Okla.  
 GOODWIN, LESLIE H., Mgr., Mine and Plant, Ely-Copperfield Associates,  
 West Fairlee, Vt.  
 GRANT, BERTRAM.....908 Fifth Ave. North, Great Falls, Mont.  
 GRAYATT, C. MARSHALL, Aviation Corps, Hayes Hall, Ohio State University,  
 Columbus, Ohio.  
 GREENE, FRED T.....Room 321, Rialto Bldg., Butte, Mont.  
 HAMILTON, HARRY T., Asst. Genl. Mgr., The Moctezuma Copper Co.,  
 Nacozari, Son., Mexico.  
 HARRIS, ARTHUR L.....La Vega, Dominican Republic.  
 HARRISON, WILLIAM S., San Francisco Mines of Mexico, Ltd., 615 Mills Bldg.,  
 El Paso, Texas.  
 HASELTINE, RICHARD S.....Parker, Ariz.  
 HAVLIN, THOMAS N.....6446 Maryland Ave., Chicago, Ill.  
 HEBERLEIN, CARL A.....Felton Hotel, Mazatlan, Sinaloa, Mexico.  
 HICKMAN, ELWOOD C.....American Smelting & Refining Co., Murray, Utah.  
 HILL, JOSEPH H., Min. Engr., Engineering Dept., Nevada Cons. Copper Co.,  
 Ruth, Nev.  
 HINCKLEY, ELMER R.....1458 Oak St., Glendale, Ariz.  
 HUNLEY, JOHN B.....Shafter, Presidio Co., Texas.  
 HU, SHIH HUNG.....Care Rev. C. P. Hu, William Nast College, Kiukiang, China.  
 HYDE, REED W.....L. H. Hyde, 502 South Eastern Ave., Joliet, Ill.  
 JAHN, WILLIAM F., Min. and Met. Engr., The Dorr Co., 17 Battery Place,  
 New York, N. Y.  
 JENSEN, JOSEPH.....Amalgamated Oil Co., Pacific Electric Bldg., Los Angeles, Cal.  
 KANE, JOHN I.....Raquette Lake, New York.  
 KEMP, DONALD C.....Eldora, Boulder Co., Colo.  
 KINNEY, HARRY D., Min. Engr.....Instructed to hold all mail.  
 KIVARI, ARTHUR M.....Hotel West Court, Denver, Colo.  
 KOEBIG, JULIUS, Cons. Chemical and Min. Engr., 934 Union Oil Bldg.,  
 Los Angeles, Cal.  
 KOMAR-OF-SKYE, GEORGE, Min. Engr., Care American Institute of Mining  
 Engineers, 29 West 39th St., New York, N. Y.  
 KONSELMAN, ALBERT S.....Globe, Ariz.  
 LABARTHE, JULES.....Bradley, Bruff & Labarthe, 85 Second St., San Francisco, Cal.  
 LAMB, MARK R.....Room 200-E, 30 Church St., New York, N. Y.  
 LANDERO, CARLOS F. DE.....580 Capel St., Oakland, Cal.  
 LATHAM, EVERETT B.....725 Merchants National Bank Bldg., Los Angeles, Cal.  
 LATHE, FRANK E., Chile Exploration Co., Chuquicamata, Chile, South America.  
 LEE, MONTROSE L.....Care S. Pearson & Son, 32 Broadway, New York, N. Y.  
 LEVIS, ALFRED C., Min. Engr.....3407 Clifton Ave., Baltimore, Md.  
 LEWIS, ROBERT S.....Box 902, Stanford University, Cal.  
 LINTON, RAYMOND A.....54 W. 40th St., New York, N. Y.  
 LOGAN, WILLIAM W.....382 50th Street, Oakland, Cal.  
 LONGERGAN, P. J., JR.....1624 Washington Ave., Colorado Springs, Colo.  
 LOO, PANG C.....308 Wood St., So. Bethlehem, Pa.  
 LORD, JAMES O.....Met., Illinois Steel Co., Gary, Ind.  
 MCCRODAN, BYRON A.....295 Front St., Belleville, Ont., Canada.  
 MAC COY, FREDERICK.....Min. Engr., Sunnyside Mining & Milling Co., Eureka, Colo.  
 MACFARLANE, A. KENNEDY, San Francisco Mines of Mexico, Ltd.,  
 615 Mills Bldg., El Paso, Texas.  
 MAGEE, JAMES F., Min. Engr.....116 East Walnut St., Lancaster, Pa.  
 MANSFIELD, MELVIN.....McGill, Nevada.  
 MATLACK, ELLWOOD V.....Terminal Hotel, St. Louis, Mo.  
 MAY, ALBERT E.....P. O. Box 1234, Amarillo, Texas.

- MENEFEE, ARTHUR B.....Union Zinc Co., Sharp's Chapel, Tenn.  
MERRICKS, HOMER I.....1414 N. E. Sixth St., Minneapolis, Minn.  
MILES, JOHN B., Expert Aid, U. S. Navy, Care Public Works Office,  
Navy Yard, Norfolk, Va.  
MORAN, ROBERT B.....632 Title Insurance Bldg., Los Angeles, Cal.  
NEWBERRY, A. W., Min. Engr.....805 Crocker Bldg., San Francisco, Cal.  
NORMAN, JOHN E.....R. F. D. No. 1, Box 15, Broomfield, Colo.  
NORTH, MORTIMER S.....Calumet Hotel, Calumet, Mich.  
PALMER, GUALTERIO C., American Smelting & Refining Co., Ave. 5 de Mayo No. 32,  
Mexico City, Mexico.  
PERRINE, IRVING.....1601 a N. Western Ave., Oklahoma City, Okla.  
PHILLIPS, WILLIAM B.....1209-10 Carter Bldg., Houston, Texas.  
PITMAN, S. M.....125 George St., Providence, R. I.  
PLACE, RICHARD G.....Aloha Club, Selby, Cal.  
RAHILLY, H. J.....Anaconda Copper Mining Co., Butte, Mont.  
RAWLINGS, STUART L., Asst. Genl. Mgr., Cerro de Pasco Copper Corp., Lima,  
Peru, South America.  
RAY, FRANK A., Cons. Mining Engr., 431 Citizens Bank Bldg., Columbus, Ohio.  
REA, JOHN E.....Ringling, Jefferson Co., Okla.  
RHOADS, ALBERT E.....1040 Woodward Ave., Detroit, Mich.  
RICHARDSON, CHARLES.....Copper Cliff, Ont., Canada.  
RIEDEL, OTTO F.....972 East 27th St., North, Portland, Ore.  
RINEHART, CHARLES R., Fuller Engineering Co., Room 2079, 50 Church St.,  
New York, N. Y.  
ROBBINS, HALLET R., Civ., Min. Engr.....Anyox, B. C., Canada  
RODGERS, C. EARL, Mgr., Davidson Gold Mines Ltd., So. Porcupine, Ont., Canada.  
ROSS, HERBERT W.....314 Pacific Ave., Piedmont, Oakland, Cal.  
ROTHSCHILD, CLARENCE G., V. Sydney Rothschild & Co., 25 Broad St.,  
New York, N. Y.  
SCHIMERKA, FRANCIS S., Research Engr., Milling Dept.,  
The Arizona Copper Co., Ltd., Morenci, Ariz.  
SCHOLZ, CARL.....547 W. Jackson Blvd., Chicago, Ill.  
SCHULTZ, ROY W., Min. & Met. Engr.....Douglass House, Houghton, Mich.  
SCOTT, ARCHIE.....Cons. Arizona Smelting Co., Humboldt, Ariz.  
SEARS, MORTIMER A.....Paintsville, Ky.  
SEIDEL, VICTOR B.....45 West 11th St., New York, N. Y.  
SHERWOOD, FRANKLIN P.....34 Pine St., New York, N. Y.  
SHONTS, SYDNEY LATHAM.....Min. Engr., Wallace, Idaho.  
SHUMWAY, RALPH W., Chief Engr., The Wolf Park Leasing Co., Canon City, Colo.  
SLATER, AMOS.....445-6 Henry Bldg., Seattle, Wash.  
SLOAN, WILLIAM A.....212 N. Louise St., Glendale, Cal.  
SMITH, HOWARD C.....Cia. de Santa Gertrudis, Pachuca, Hgo, Mexico.  
SMITH, JAMES H., JR.....Reserve Officers Training Camp, Fort Niagara, N. Y.  
STARKWEATHER, HERBERT.....Route 2, Edgewater, Colo.  
STAYER, WILLIAM H., Cidade de Bomfim, Estado de Bahia, Brazil, South America.  
STEIDLE, EDWARD.....Min. Engr., U. S. Bureau of Mines, Car No. 1, Reno, Nev.  
STEWART, ROBERT H., Mining Engr., 736 Granville St., Vancouver Block,  
Vancouver, B. C., Canada.  
STONE, GALEN L.....Banker, Hayden, Stone & Co., 87 Milk St., Boston, Mass.  
STOTZ, NORMAN I.....433 Warren Ave., W., Detroit, Mich.  
SUTTON, HARRY A.....Aumsville, Marion Co., Ore.  
SWAIN, SETH R., Met. Engr.....Box 117, East Orange, N. J.  
TAGGART, OLIVER R., Head Chemist, St. Louis Smelting & Refining Co.,  
Collinsville, Ill.  
THOMAS, DAVID R.....Sellwood, Ont., Canada.  
THOMPSON, PERCY W.....Standard Oil Co., Los Alamos, Cal.  
THROCKMORTON, HOWARD W.....441 California St., San Francisco, Cal.  
TYLER, PAUL M.....Min. Engr., Ely Copperfield Associates, West Fairlee, Vt.  
VAN DORP, GLENN H.....1115 Polk St., Topeka, Kans.  
VISEL, CLIFTON E.....49 Morse St., New Haven, Conn.  
VRANG, CHRISTIAN.....S. P. Geol. Dept., Flood Bldg., San Francisco, Cal.  
WAGNER, WILLIAM HUFF, Gen. Mgr. The Copper King Mine Produ'cs Co.  
Copperton, via McCoy, Eagle Co., Col.  
WALLACE, H. VINCENT, Min. Engr.,.....620 N. Warren St., Helena, Mont.  
WALTER, E. W.....Silverton, Colo.  
WARD, ARTHUR T.....Bellefonte, Pa.

|  |   |
|--|---|
| WASSON, H. J.  | Prescott, Ariz.                                     |
| WEPPER, GOTTLIEB W.  | 1915 Virginia Street, Berkeley, Cal.                |
| WESTER, JAMES R.   | Box 1401, Bisbee, Ariz.                             |
| WHEELER, H. A.   | 408 Locust St., St. Louis, Mo.                      |
| WILLIAMS, CHARLES F.   | 25 Blymyer Ave., Mansfield, Ohio.                   |
| WILLIAMS, JAMES M., JR.  | Continental, Pima County, Ariz.                     |
| WILLISTON, GEORGE F., Min. Engr.   | Hualapai Metals Co., Box 827, Kingman, Ariz.        |
| WILMOT, H. CLIFFORD, Ajax-Hocking Coal Co.,                                  | 441 Title Bldg., Baltimore, Md.                     |
| WINCHELL, JOHN H., JR.   | Engr., Demming Mines Co., Murphy, Idaho.            |
| WITT, HERBERT N.   | 805 Crocker Bldg., San Francisco, Cal.              |
| WITTENAU, ERNEST, Asst. Supt. of Concentration,                              | The Arizona Copper Co., Ltd.,<br>Morenci, Ariz.     |
| WOOD, FLETCHER H.  | De Soto Mine, Ocotillo, Ariz.                       |
| WOOD, JAY P.   | The Utah Metal & Tunnel Co., Bingham Canyon, Utah.  |
| WRIGHT, PERCY E., Cons. Mech. Engr.  | 2012 L. C. Smith Bldg., Seattle, Wash.              |
| WYER, SAMUEL S., Consulting Engr.  | 1014 Hartman Bldg., Columbus, Ohio.                 |
| YEANDLE, WILLIAM H., JR., U. S. Smelting, Refining & Mining Exploration Co., | 1504 Hobart Bldg., San Francisco, Cal.              |
| YOUNG, HORACE G.   | Engineers Training Depot, St. Johns, P. Q., Canada. |

## MEMBERS' ADDRESSES WANTED

| Name.  | Last address of Record from which Mail has been returned.   |
|--|---|
| BONILLAS, Y. S.                                    | P. O. Box 2014, Bisbee, Ariz.                               |
| CASTLETON, WILLIAM A.                              | Canyon Ferry, Mont.   |
| DENNIS, ARTHUR C.                                  | Gulf Production Co., 915 North Dixon St., Gainesville, Tex. |
| HOOKE, WALTER, 75 Parliament Hill Mansions,        | Gospel Oak, Highgate N.,<br>London, England.                |
| KERR, DUNCAN M.                                    | Winthrop, Cal.  |
| KLEINSCHMIDT, FRANZ H., Blue Jacket & Queen Mines, | Landore, Adams Co., Idaho.                                  |
| LOWE, HENRY P.                                     | Central City, Colo.   |
| MCHARDY, ROY H.                                    | Arizona Cons. Mining Co., Globe, Ariz.                      |
| RAY, JAMES C.                                      | Oatman, Ariz.   |
| RECKNAGEL, RUDOLF.                                 | 13 Austin Friars, London, E. C., England.                   |
| SCHNEPP, CHARLES F.                                | Riverside Club, Penns Grove, N. J.                          |
| TONG, SING K.                                      | 404 West 115th St., New York, N. Y.                         |

## NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period May 10 to June 10, 1917.

| Date of Election. | Name.                | Date of Death. |
|-------------------|----------------------|----------------|
| 1916              | Chen, H. T.          | — — 1917.      |
| 1903              | Day, Arthur M.       | May 1, 1917.   |
| 1915              | Gheur, Ernest        | May 3, 1917.   |
| 1887              | Gordon, Henry A.     | Apr. 30, 1917. |
| 1892              | Hughes, A. D.        | Apr. 14, 1917. |
| 1889              | Locke, Franklin D.   | May 11, 1917.  |
| 1887              | Mackenzie, George S. | Dec. — 1916.   |
| 1882              | Moore, Charles J.    | Jan. — 1917.   |
| 1903              | Perry, Orville C.    | May 9, 1917.   |

# BIOGRAPHICAL NOTICE OF H. T. CHEN

We note with sincere regret the death of H. T. Chen, who was killed by forest fire in the mountainous region of Koku-Chang where he had been operating as Engineer in Chief of the Yunnan Tin Trading Co. Mr. Chen graduated in 1911 from the Department of Mines, University of California, being the first Chinese Government scholar in mining to attend that institution, and one of the first in America. He came of a distinguished family, many members of which are in official life in Peking and in his native province of Hunan, all of whom are noted for their classical scholarship. He was himself a Hsiao-tsai or Bachelor of Arts, a coveted degree of the old literati class, which he won under the Manchu regime when he was still a stripling. His mining work in Yunnan was carried on with only the most primitive facilities and in a distant province, but was, nevertheless, attended with great success and profit for his company.

The region is infested with brigands and rebellious troops who frequently turn marauders and prey upon the countryside, and Mr. Chen has had to go to the mines always with a large armed guard sometimes aggregating 150 men and some hundreds of miners, also armed. Two armed squads of men keep in touch with his base at Koku-Chang every day to get supplies and food, and it is feared that the brigands who have thus been prevented from plundering and robbing his parties have started the fires to burn him out, with the disastrous result reported. Mr. Chen has thus given his life to his country in the attempt to win for modern China and Chinese civilization the raw materials of her new industrial life. He did so cheerfully and with full appreciation of the dangers that beset him, and China regards this as one of the most noble sacrifices of her young sons on the altar of patriotism. He is survived by a daughter of two years and by his widow who was also a graduate of the University of California and will be remembered by her classmates as Lucy Kong, 1912.

A friend of Mr. Chen's writes as follows:

"The fate and achievements of this pioneer mining engineer should be of interest to members of the American Institute of Mining Engineers and should also serve as an inspiration to the large number of young men who are preparing to enter the profession of mining engineer. It will be well for them to read of this example of large-hearted and unselfish devotion to the cause of pushing back the wilds of nature all for the sake of China's industrial regeneration, a cause to which he dedicated his life long before he left China for scientific education in America, and a cause which he never forgot and served faithfully to the end of his life."



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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Influence of Base Metals in Gold Bullion Assaying\*

BY FREDERIC P. DEWEY,† PH. B., WASHINGTON, D. C.

HAVING shown<sup>1</sup> the difficulty of assaying so-called cyanide bullion and the extreme variations often found in the results, an investigation was undertaken to discover if possible the causes of these variations by a series of tests upon the specific action of various metals during cupellation.

**Zinc.**—In order to test the effect of zinc, 48 assays were made in sets of 6, 2 rows of 3. The charge for the middle cupels consisted of 500+ mg. Au, 1,125 mg. Ag, 10 mg. Cu. To the end cupels 50, 75, 100 and 125 mg. Zn were added, and a set of each proportion was cupelled with 4 and 8 grams Pb. The cupels were assayed and showed the following gold contents in milligrams, the figures being arranged as the cupels stood in the muffle:

TABLE 1

|                   | 50 Mg. Zn |      |      | 75 Mg. Zn |      |      | 100 Mg. Zn |      |      | 125 Mg. Zn |      |      |
|-------------------|-----------|------|------|-----------|------|------|------------|------|------|------------|------|------|
| 4 grams, Pb ..... | 0.28      | 0.28 | 0.28 | 0.29      | 0.27 | 0.34 | 0.33       | 0.28 | 0.41 | 0.31       | 0.30 | 0.36 |
|                   | 0.36      | 0.33 | 0.34 | 0.31      | 0.31 | 0.29 | 0.35       | 0.37 | 0.42 | 0.41       | 0.34 | 0.38 |
| 8 grams, Pb ..... | 0.46      | 0.43 | 0.42 | 0.38      | 0.36 | 0.43 | 0.35       | 0.34 | 0.41 | 0.33       | 0.37 | 0.37 |
|                   | 0.54      | 0.47 | 0.59 | 0.44      | 0.43 | 0.42 | 0.38       | 0.39 | 0.41 | 0.57       | 0.38 | 0.46 |

In many instances the zinc does not appear to affect the cupel gold, but various exceptions are apparent, generally in the cupels in the back row. With 100 and 125 mg. Zn, in some cases the Zn burned with its characteristic flame before it could alloy with the other metals. Also, less fusible oxidized lumps were observed during the cupellation and some cupels showed a scum.

In order to lessen the premature burning out of the Zn, an alloy of 2 parts Ag to 1 part Zn was used. A uniform charge of 500+ mg. Au,

\* Published by permission of the Director of the Mint.

† Assayer, Bureau of the Mint.

<sup>1</sup> Assay and Valuation of Gold Bullion, *Trans.* (1909), 40, 780-797.

Annual Report of the Director of the Mint, 1909, 25-36.

Sampling of Gold Bullion, *Trans.* (1912), 44, 853-879.

Treasury Department Document 2672.

Cadmium and Nickel in Cyanide Bullion, *Engineering & Mining Journal* (Apr. 13, 1912), 93, 733.

300 mg. alloy (containing 100 mg. Zn), 925 mg. Ag and 20 mg. Cu was used in sets of 9, 3 rows of 3. One set each was run with 4, 5, 8 and 10 grams Pb. In the table the first column gives the relation of the cornet weight to the amount of Au taken and the second the amount of Au found by assaying the cupel, including beads, as well as Au actually absorbed by the cupel.

TABLE 2

| 4 Grams Pb  |            | 5 Grams Pb  |            | 8 Grams Pb  |            | 10 Grams Pb |            |
|-------------|------------|-------------|------------|-------------|------------|-------------|------------|
| Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. |
| 0.12+       | 0.12       | 0.35+       | 0.18       | 0.58—       | 1.09       | 0.08+       | 0.31       |
| 0.29+       | 0.20       | 0.41+       | 0.19       | 0.14+       | 0.36       | 0.11+       | 0.28       |
| 0.15—       | 0.48       | 0.16+       | ....       | 0.25+       | 0.28       | 0.11+       | 0.34       |
| 0.22+       | 0.35       | 0.22+       | 0.37       | 0.18+       | 0.38       | 0.34—       | 0.82       |
| 0.32+       | 0.24       | 0.34+       | 0.27       | 0.08+       | 0.47       | 0.02+       | 0.48       |
| 0.33+       | 0.23       | 0.16—       | 0.54       | 0.19+       | 0.35       | 0.06—       | 0.50       |
| 0.02—       | 0.34       | 0.25+       | 0.27       | 0.03—       | 0.45       | 0.04+       | 0.51       |
| 0.25+       | 0.26       | 0.30+       | 0.32       | 0.04—       | 0.54       | 0.14—       | 0.60       |
| 0.23+       | 0.24       | 1.79—       | 2.43       | 0.09+       | 0.43       | 0.11—       | 0.50       |

In a second set 500 mg. of the alloy, containing 166 mg. Zn and 790 Ag, were used, giving the following:

| 4 Grams Pb  |            | 5 Grams Pb  |            | 8 Grams Pb  |            | 10 Grams Pb |            |
|-------------|------------|-------------|------------|-------------|------------|-------------|------------|
| Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. |
| 4.80—       | 5.01       | 1.97—       | 2.10       | 0.06+       | 0.37       | 0.07+       | 0.30       |
| 0.31+       | 0.14       | 0.19+       | 0.24       | 0.25+       | 0.38       | 0.04+       | 0.34       |
| 1.80—       | 2.17       | 0.01—       | 0.42       | 0.23—       | 0.75       | 0.21—       | 0.69       |
| 0.21—       | 0.62       | 0.42—       | ....       | 0.17+       | 0.39       | 0.24—       | 0.64       |
| 0.25+       | 0.25       | 0.19+       | 0.28       | 0.17+       | 0.42       | 0.13—       | 0.44       |
| 0.39—       | 1.02       | 4.08—       | 4.46       | 0.28—       | 0.41       | 2.04—       | 2.44       |
| 1.32—       | 1.70       | 0.05—       | 0.34       | 0.51+       | 0.46       | 0.34—       | 0.79       |
| 0.19+       | 0.22       | 0.10+       | 0.35       | 0.13—       | 0.51       | 0.18—       | 0.49       |
| 4.41—       | 4.63       | .....       | 5.71       | 0.10+       | 0.48       | 0.44—       | 0.93       |

In these tests the pyrometer registered about 950° C. The figures are arranged as the cupels stood in the furnace.

From these figures, it is seen that Zn may be a most disturbing element. In various cases more or less infusible oxidized lumps were observed floating on the bead and these lumps entangled metal. In all probability this was due to the extreme preferential solubility of Au and Ag in Zn, the immiscibility of the resulting alloy with Pb, and its lower specific gravity; from all of which the Zn-Au-Ag alloy was formed, came

to the surface of the bead and the Zn was oxidized without sufficient Pb to liquefy the ZnO. Eventually some of these lumps inclosing Au and Ag reached the cupel.

In a final comparison the Ag-Zn alloy was used to introduce small amounts of Zn, three proportions—62.5, 85 and 125 mg. of alloy—being used. Each proportion was cupelled with 4, 5, 8 and 10 grams Pb, in sets of 9 cupels, 3 rows of 3, the middle cupel charge being without Zn as in the first test. This gives 36 rows, but 2 rows were lost. The 68 cupels with Zn yielded 21.39 mg. Au, and the 34 without, 10.70 or substantially half as much, thus showing that small amounts of alloyed Zn has no effect upon the cupel gold. However, the ZnO was not always properly absorbed by the cupel. With 125 mg. alloy there was a distinct tendency for the Zn-Au-Ag alloy to come to the surface. Curiously, this was also shown with the small amounts of alloy when 10 grams Pb were used. Evidently, the longer time required to oxidize such amounts of Pb offered sufficient opportunity for the Zn-Au-Ag alloy to form and gather at the surface. Therefore, the use of larger amounts of Pb to slag off Zn is not advisable.

**Cadmium.**—Having shown<sup>2</sup> that Cd is often present in cyanide bullion, a few sets of 2 rows of 3 cupels were run, replacing the Ag-Zn alloy by Ag-Cd. One-half of the results obtained with each amount of Pb and Cd are given. The other results were similar.

TABLE 3

| 4 Grams Pb, 166 Mg. Cd |            | 10 Grams Pb, 100 Mg. Cd |            |
|------------------------|------------|-------------------------|------------|
| Cornet, Mg.            | Cupel, Mg. | Cornet, Mg.             | Cupel, Mg. |
| 0.60+                  | 0.19       | 0.10+                   | 0.38       |
| 0.57+                  | 0.17       | 0.09+                   | 0.38       |
| 0.60+                  | 0.16       | 0.14+                   | 0.38       |
| 0.58+                  | 0.15       | 0.08+                   | 0.44       |
| 0.64+                  | 0.18       | 0.03+                   | 0.48       |
| 0.58+                  | 0.19       | 0.04+                   | 0.44       |

Aside from four beaded cupels, the results were uniform and very satisfactory, showing that Cd cannot be the cause of the wide variation so often found in cyanide bullion assays. No infusible lumps were observed in the cupelling beads, but in the 166 mg. Cd, 4 Pb sets, the cupels were heavily coated with the characteristic dark brown oxide of Cd.

The low Zn alloy runs were duplicated with the Cd alloy, and a most surprising result was obtained. Here also 2 rows were lost. The 68 Cd cupels yielded 18.14 mg. Au, and the 34 without Cd yielded 9.28 mg. Au. This is 0.21 mg. more than half of the yield of the Cd cupels and shows a distinct protective action by the Cd.

<sup>2</sup> *Engineering and Mining Journal*, op. cit.

Further evidence of this protective action of Cd is furnished by a test in which an alloy of Cd-Zn-Cu-Ag was used. The charge in the end cupels was 500+ mg. Au, 500 mg. alloy, 740 mg. Ag. This gave approximately 40 mg. each of Cd and Zn and 25 mg. Cu. The middle cupels held 500+ mg. Au, 1,125 mg. Ag, 20 mg. Cu. These charges were run in sets of 9, 3 rows of 3, with 4, 5, 8 and 10 grams of Pb. In every row except one, the middle cupel yielded more gold than either end cupel. In the exception, the middle cupel yielded 0.06 mg. more than one end, and 0.02 less than the other, or 0.02 mg. more than the average of the ends. The 24 end cupels yielded 10.42 mg. Au, and the 12 middle ones 5.89, which is 0.68 mg. more than half of the ends. The following table, with 4 Pb, is characteristic, all the cornets being plus:

|             | First Row |      |       | Second Row |      |      | Third Row |      |      |
|-------------|-----------|------|-------|------------|------|------|-----------|------|------|
| Cornet..... | 0.45      | 0.30 | 0.30. | 0.39       | 0.28 | 0.33 | 0.34      | 0.18 | 0.35 |
| Cupel.....  | 0.25      | 0.31 | 0.28  | 0.29       | 0.34 | 0.32 | 0.31      | 0.39 | 0.29 |

In some miscellaneous tests, 16 Cd end cupels yielded 3.58 mg. Au, and the 8 center ones, without Cd, 2.21 mg., 0.42 mg. more than half of the ends.

In many of the Cd cupellations the characteristic dark brown oxide was shown.

*Copper.*—An alloy of 8 parts Ag, 2 parts Cu, designated as 800 Ag, was used. The assay consisted of 450+ mg. Au, and 1,250 mg. Ag-Cu alloy. They were run with 5 and 10 grams Pb at two temperatures, stated as high and low, in 3 rows of 3 cupels.

TABLE 4 (First part)

| 5 Grams Pb  |            |             |            |
|-------------|------------|-------------|------------|
| Temp. Low   |            | Temp. High  |            |
| Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. |
| 0.44—       | 0.29       | 4.31—       |            |
| 0.13—       | 0.26       | 4.19—       | 3.66       |
| 0.28—       | 0.32       | 4.22—       | 3.81       |
| 0.62—       | 0.37       | 2.16—       | 1.23       |
| 1.00—       | 0.26       | 2.02—       | 1.93       |
| 0.88—       | 0.27       | 1.58—       | 1.74       |
| 0.98—       | 0.43       | 1.54—       | 1.48       |
| 1.16—       | 0.49       | 1.69—       | 1.70       |
| 1.23—       | 0.66       | 1.67—       | 1.89       |

The figures in Table 4 call for further and more elaborate investigation, but time was lacking to continue this work.

When Cu is present in bullion we generally use more Pb and a higher temperature in cupellation. Both these conditions increase the cupel

TABLE 4 (Second part)

| 10 Grams Pb |            |             |            |
|-------------|------------|-------------|------------|
| Temp. Low   |            | Temp. High  |            |
| Cornet, Mg. | Cupel, Mg. | Cornet, Mg. | Cupel, Mg. |
| 0.05+       | 0.25       | 2.47—       | 2.89       |
| 0.04+       | 0.23       | 2.93—       | 3.16       |
| 0.10—       | 0.19       | 2.36—       | 2.79       |
| 0.08—       | 0.28       | 2.79—       | 3.27       |
| 0.03+       | 0.32       | 3.15—       | 3.55       |
| 0.18—       | 0.29       | 3.31—       | 3.57       |
| 0.14—       | ....       | 3.20—       | 3.56       |
| 0.01—       | 0.32       | 3.17—       | 3.51       |
| 0.91—*      | 1.16*      | 3.03—       | 3.43       |

\*Bead in cupel.

absorption of Au. In order to ascertain the effect of Cu *per se* upon the cupel absorption, 8 sets of 6 cupels, 2 rows of 3, were run with 5 to 7 mg. Cu, and 8 were run with 50 mg. Cu. Great care was exercised to have all of the other conditions as nearly as possible the same in every case, but a detailed inspection of the results indicates that the attempt was not altogether successful. The general temperature was shown by a pyrometer, and kept uniform, but the results show that the temperature of the beads did not conform to the pyrometer and varied considerably.

The cupel absorptions on the low Cu varied from 0.23 to 0.33 mg., and on the high Cu from 0.24 to 0.34 mg., but there was a very decided preponderance of low absorptions with low Cu and high ones with high Cu. The average absorptions were:

|                 |        |
|-----------------|--------|
|                 | Mg.    |
| High Cu.....    | 0.2991 |
| Low Cu.....     | 0.2773 |
| Difference..... | 0.0218 |

In order to keep the conditions the same, more Pb and a higher temperature were used on the low Cu, than were necessary, but the test clearly shows that Cu increases the cupel absorption.

The same fact is also shown in a general way by 26 rows with Cu in the ends, but without Cu in the centers. The Cu was alloyed with Ag and 2 proportions of Cu, 62.5 mg. and 125 mg. were each used with 4, 5, 8 and 10 grams Pb. The end cupels yielded 9.33 mg. Au, and the centers 4.12, or 0.545 mg., less than half of the ends.

**Lead.**—Being the universal constituent of cupelling beads, the action of this metal is of the utmost importance. Many results have been published regarding its effects, but they are not, and could not really be expected to be, harmonious. There are so many variables involved that

a tremendous amount of most painstaking work would be required to reach a quantitative expression of the relation of the weight of Pb in a bead to its effect upon the assay. I give here only a few illustrations, proposing to deal with the subject in detail in a separate paper on cupellation.

Under one set of conditions, 3 sets of 9 assays of fine gold showed the following cupel absorptions of Au, with differing amounts of Pb:

TABLE 5

| Pb, 0.5 Gram |        | Pb, 2 Grams |        | Pb, 5 Grams |        |
|--------------|--------|-------------|--------|-------------|--------|
| Mg.          | Cupels | Mg.         | Cupels | Mg.         | Cupels |
| 0.06         | 2      | 0.20        | 1      | 0.53        | 1      |
| 0.07         | 4      | 0.21        | 4      | 0.54        | 1      |
| 0.10         | 1      | 0.22        | 3      | 0.55        | 2      |
| 0.12         | 1      | 0.26        | 1      | 0.56        | 2      |
| 0.15         | 1      | ....        | ...    | 0.58        | 1      |
|              | <hr/>  |             | <hr/>  | 0.60        | 1      |
|              | 9      |             | 9      | 0.64        | 1      |
|              |        |             |        |             | <hr/>  |
|              |        |             |        |             | 9      |

However, under a different set of conditions, and with 2 grams Pb, 8 sets of 6 assays of fine gold in one cupel gave 0.05 mg. absorption in one cupel, 0.06 mg. in 7 cupels, 0.07 mg. in 15 cupels, 0.08 mg. in 18 cupels, 0.09 mg. in 5 cupels and 0.11 in 2 cupels, total 48 cupels. Five sets of 6 assays in another cupel gave a preponderance of absorptions between 0.07 and 0.10 mg.

Even with 4 grams Pb, and on coin gold (900 Au, 100 Cu), the absorptions were less in another laboratory than they were on fine gold and 2 grams Pb in the first test.

Speaking in a very general way, under one set of conditions, doubling the Pb or increasing the pyrometer reading by 100° doubled the Au absorption, while both doubling the Pb and raising the pyrometer reading by 100° quadrupled the absorption of Au.

*Silver.*—Intimately connected with the action of base metals is the protective action of Ag upon the absorption of Au by the cupel. Here, again, much work is required to reach quantitative results, but the general effect is shown by the cupel absorption of Au in the base metal assay and the corresponding Au assay.

Various samples were taken from a mass melt weighing 774.3 oz., made from 33 bars. On these 16 Au and 16 base assays were made. The absorptions in the gold assays varied between 0.17 and 0.27 mg., with the average 0.21 mg. The absorptions in the base assays varied between 0.64 and 1.00 mg. with the average 0.81 mg.

As a direct test of this question, sets of 6 cupels, 2 rows of 3, were run with varying proportions of Ag, and at 3 pyrometer readings. The total amount of gold absorbed in each set, and the average per cupel, were, in milligrams:

TABLE 6

| Parts of Ag |                    | Pyrometer Readings |         |           |
|-------------|--------------------|--------------------|---------|-----------|
|             |                    | 900° C.            | 950° C. | 1,000° C. |
| 3           |                    | 0.290              | 0.350   | 0.490     |
|             |                    | 0.250              | 0.420   | 0.520     |
|             |                    | 0.240              | 0.430   | 0.730     |
|             |                    | 0.250              | 0.400   | 0.700     |
|             | Av. per cupel..... | 0.043              | 0.066   | 0.101     |
| 2½          |                    | 0.380              | 0.530   | 0.750     |
|             |                    | 0.340              | 0.490   | 0.790     |
|             |                    | 0.260              | 0.550   | 0.770     |
|             |                    | 0.300              | 0.610   | 0.740     |
|             | Av. per cupel..... | 0.053              | 0.091   | 0.127     |
| 2¼          |                    | 0.440              | .....   | 0.780     |
|             |                    | 0.450              | .....   | 0.880     |
|             |                    | 0.470              | .....   | 0.820     |
|             |                    | 0.470              | .....   | 0.830     |
|             | Av. per cupel..... | 0.076              | .....   | 0.138     |
| 2           |                    | 0.570              | 0.620   | 0.920     |
|             |                    | 0.520              | 0.660   | 0.830     |
|             |                    | 0.630              | 0.930   | 0.970     |
|             |                    | 0.700              | 0.820   | 0.930     |
|             | Av. per cupel..... | 0.101              | 0.126   | 0.152     |

Under practical every-day conditions, the protection afforded by the Ag within ordinary parting conditions may be easily overbalanced by differences in other conditions.

I am greatly indebted to a former assistant, Mr. Scott Carter, for much painstaking effort in working out the data of this investigation.

#### Conclusions

In small amounts, Zn alone has no effect upon the cupel Au. In large amounts the Zn-Au-Ag alloy is formed, comes to the surface and carries Au to the cupel. A tendency to do this occurs when small amounts of Zn are cupelled with large amounts of Pb.

In all the comparative tests Cd showed a distinct protection of the Au, even when Zn was also present.

The presence of Cu *per se* directly increases the absorption of gold, but its effect is intensified by the higher temperature and larger amount of Pb required.

The effect of Pb varies greatly with the surrounding conditions.

The presence of Ag is a good protection to the Au, but may be over-balanced by other conditions.

### Addenda

The following list of references on this subject has been prepared by the Library Service Bureau of the United Engineering Society. It was thought that these references were of sufficient interest to be included:

- RÖSSLER, H.: Investigations on the Assay of Gold as Applied to Gold Coins. *Dingler's Polytechn. Jour.* (1872), **206**, 188. Influence of lead on the cupellation of gold-bearing products.
- FURMAN, H. VAN F.: Losses of Gold and Silver in the Fire-Assay. *Trans.* (1894), **24**, 735. Amount of base metals in lead-button considered as one of the various conditions affecting losses during cupellation. Zinc appears to be the most objectionable element. Copper also increases the losses.
- FURMAN, H. VAN F.: *A Manual of Practical Assaying*, 1903, 5th ed., New York. pp. 386-400. Losses of gold and silver in the fire-assay.
- LODGE, R. W.: *Notes on Assaying and Metallurgical Laboratory Experiments*, 1904, New York, John Wiley & Sons, 143-144. Table showing effect of copper on cupellation of gold, with lead, gold and temperature remaining constant.
- ROSE, T. K.: On Cupellation and Parting in Ore Assaying. *Jour. Chem. Met. Soc. South Africa* (1905, January), **5**, 167. Result of some experiments on the losses in cupellation due to the presence of various impurities in the lead buttons are given in tabulated form. In each case the charge consisted of 1 mg. of gold, 4 mg. of silver and 1 gram. of the impurity with 25 grams of lead.
- FULTON, CH. H.: *A Manual of Fire Assaying*, 1907, Hill Publishing Co., pp. 71-80, particularly Table XIV. Copper in cupellation of silver and gold, pp. 121-123. Losses in assay for gold and silver due to amount of lead with which gold and silver is cupelled, and the nature and amount of impurities present.
- SMITH, E. A.: Assay of Industrial Gold Alloys. *Jour. Inst. Metals* (1910), **3**, 98-123. Effect of various properties of lead for cupellation of gold-copper alloys extensively discussed. Results of experiments given in tabulated form.
- SMITH, E. A.: *Influence of Base Metals on Cupellation*. Sampling and assay of the precious metals: comprising gold, silver, platinum, and the platinum group metals in ores, bullion and products. London, 1913, Ch. Griffin, pp. 167-176. Complete critical review of the literature on this subject, together with some results obtained by the author himself (Table XVII, p. 173).
- DEWEY, F. P.: Determination of Silver and Base Metal in Precious Metal Bullion. *Jour. Ind. Engng. Chem.* (1914), **6**, 650-658; 728-736.
- ROSE, SIR T. K.: *The Metallurgy of Gold*, 6th ed., London, 1915, pp. 507-509. Influences of base metals on cupellation.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 39 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Relation of Sulphur to Variation in the Gravity of California Petroleum\*

BY G. SHERBURNE ROGERS,† PH. D., WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

### *Introduction*

ONE of the features of oil-field work that puzzles operator, chemist and geologist alike, is variation in the gravity of the petroleum produced on neighboring leases or even from adjoining wells. Few fields in the United States yield oil that is of uniform gravity throughout; in some fields the different grades are produced from different sands, in a few from different formations, and in many from different areas in the same sand. In a few cases, perhaps, a common assumption—that the heavier oil is derived from the lighter through the escape of the more volatile constituents—is valid, but many cases can not be explained in this way. Although in the California fields the specific gravity of the oil generally decreases with depth, there are numerous exceptions to this rule; and, as in many other regions the light oil is found nearest the surface, it is evident that evaporation underground is generally an unimportant factor.

It is generally recognized that the great differences between petroleum from widely separated fields are due largely to differences in age and degree of metamorphism and to variation in the composition of the original organic matter; but in the writer's opinion some of these broader differences, as well as most of the minor variations in the character of the oil from any one field, are due to the local action of natural reagents such as sulphur or oxygen. This is particularly likely to be true if the oil has migrated from one formation to another, as in the California fields; and in some areas it appears that variation in sulphur content may even afford a rough index of the extent or distance of the migration. It is the purpose of this paper to call attention to the chemical action of sulphur and oxygen on petroleum, to show that the gravity of the oil from any one field is intimately related to its sulphur content, and to discuss the possi-

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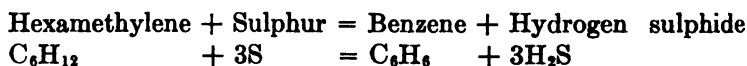
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bility that oil in its migration will encounter and take up sulphur or oxygen.

### *Chemical Action of Sulphur and Oxygen on Petroleum*

Petroleum chemists have been occupied so largely in working out the complex constitution of petroleum and the processes of refining it, that we know little of the effect on petroleum of any of the substances with which it comes in contact underground. The study of petroleum from the geochemical standpoint has received slight attention except from a few chemists, and it is therefore impossible to trace in detail the natural changes that oil may undergo. Clifford Richardson,<sup>1</sup> however, pointed out many years ago the influence of sulphur in hardening asphalt. The general action of sulphur and oxygen on oil has long been known to chemists, but the reactions involved have apparently never been exhaustively studied and their geochemical significance does not seem to have been appreciated by geologists.

If paraffin, or a paraffin-bearing oil, is digested with sulphur at moderate temperature it becomes black and heavy and finally passes to a substance resembling solid asphalt. Similarly, if a light asphaltic oil is treated with sulphur it also becomes darker and more viscous, finally becoming asphalt. Under laboratory conditions the reaction is accompanied by the evolution of hydrogen sulphide, and in fact an old laboratory method of generating hydrogen sulphide consists in heating paraffin and sulphur in a retort. The sulphur atom, by extracting 2 atoms of hydrogen from the oil, causes a condensation or polymerization of the hydrocarbon molecule, and this change is reflected in the increase of the gravity of the oil itself as it approaches solid asphalt. A simple example of change through the action of sulphur, from the polymethylene series of the general formula  $C_nH_{2n}$ , to the heavier aromatic series of the general formula  $C_nH_{2n-6}$ , may be written:<sup>2</sup>



This type of reaction is taken advantage of in the Dubb<sup>3</sup> process for manufacturing asphalt commercially by heating crude petroleum or petroleum residue with sulphur. A variety of artificial asphalt known as Pittsburgh flux, made by treating with sulphur the residuum of Pennsylvania petroleum, is described as tough and sticky, and melting only at high temperature. According to Richardson<sup>4</sup> the condensation of one of

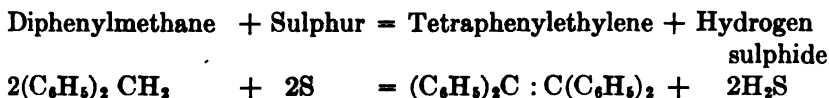
<sup>1</sup> C. Richardson: On the Nature and Origin of Asphalt. *Journal of the Society of Chemical Industry* (1898), 17, 28.

<sup>2</sup> H. Köhler: *Die Chemie und Technologie der natürlichen und künstlichen Asphalte*, p. 112, Brns., 1904.

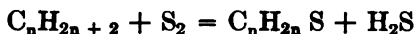
<sup>3</sup> English patent 3026, Feb. 16, 1892.

<sup>4</sup> C. Richardson: *Journal of the Society of Chemical Industry* (1897), 16, 125.

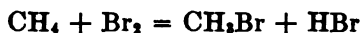
the hydrocarbon molecules involved in this process may be represented as follows:



Although the dehydrogenation or condensation of the hydrocarbon molecule caused by the formation of  $\text{H}_2\text{S}$  is the change of greatest interest geologically, under most conditions some of the sulphur enters into combination with certain oil constituents to form simple sulphur compounds or complex sulphur derivatives. Nearly all petroleum contains sulphur, and the nature of some of its simpler compounds has been investigated. In the Lima oil of Ohio, Mabery and Smith<sup>5</sup> found normal sulphides of the paraffin series, having the general formula  $\text{C}_n\text{H}_{2n} + 2\text{S}$  and ranging from  $\text{C}_2\text{H}_6\text{S}$  to  $\text{C}_{12}\text{H}_{26}\text{S}$ . In Canadian petroleum, Mabery and Quayle<sup>6</sup> isolated another series of sulphur compounds of the general formula  $\text{C}_n\text{H}_{2n}\text{S}$ . Eight members of this series, which were called thiophanes, were described. According to Köhler,<sup>7</sup> such compounds may be formed by the reaction



which may be regarded as analogous to ordinary substitution by halogens:



Sulphur is readily soluble in petroleum and some of the Texas oil contains both hydrogen sulphide and free sulphur in solution, in addition to the compounds of sulphur. According to Mabery the proportion of free sulphur in the Humble crude approaches the saturation point, so that sulphur frequently separates out of this oil during transportation.<sup>8</sup> Apparently the solution of sulphur in the heavier hydrocarbons of petroleum is immediate, even at ordinary temperature,<sup>9</sup> and in Peckham's

<sup>5</sup> C. F. Mabery, and A. W. Smith: The Sulphur Compounds in Ohio Petroleum, *American Chemical Journal* (1891) 13, 233.

<sup>6</sup> C. F. Mabery, and W. O. Quayle: The Sulphur Compounds and Unsaturated Hydrocarbons in Canadian Petroleum, *Proceedings of the American Academy of Arts and Sciences* (1905), 41, 89.

<sup>7</sup> *Op. cit.*, 74.

<sup>8</sup> C. F. Mabery: Relations of the Chemical Composition of Petroleum to Its Genesis and Geologic Occurrence, *Economic Geology* (1916), 11, 520. Also F. C. Thiele: Free Sulphur in Petroleum from Beaumont, *Journal of the Society of Chemical Industry* (1902), 21, 1271.

<sup>9</sup> The solubility of sulphur in oil naturally varies according to the series of hydrocarbons involved. Thus, the Edeleanu process for refining petroleum by liquefied sulphur dioxide depends on the fact that the latter is taken up by aromatic hydrocarbons even at very low temperatures, but by the paraffins only at higher temperatures. L. Edeleanu: *Trans.* (1914), 50, 809-827.

opinion the reaction between sulphur and the hydrocarbons, whereby the latter are rendered heavier and more asphaltic, also proceeds at ordinary temperature.<sup>10</sup> The continuous evolution of large quantities of  $\text{H}_2\text{S}$  from the asphalt lake at Trinidad may be regarded as supporting this statement. Finally, according to Endemann,<sup>11</sup> heavy hydrocarbons react with pyrite just as with free sulphur, half of the sulphur in the pyrite molecule ( $\text{FeS}_2$ ) being given off as  $\text{H}_2\text{S}$  and the rest remaining as a lower sulphide of iron.

Oxygen also exercises a pronounced effect on petroleum and the general character of its action is commonly known. When oil is exposed to the air for a long time it becomes dark, heavy and viscous, and finally passes to asphalt. This change is due chiefly to the evaporation of the more volatile constituents, but partly to oxidation. Thus, if hot air is passed through oil for several hours the oil becomes black and asphaltic.<sup>12</sup> The action of oxygen is similar to that of sulphur; some of the oxygen may be taken into combination with the oil to form complex acids or phenols, and part of it abstracts hydrogen from the hydrocarbon molecule to form water. This reaction is involved in the Byerly process for preparing asphalt by slowly distilling petroleum while passing air through it.<sup>13</sup> Oxygen, like sulphur, may also cause direct polymerization; Hausmann and Pilat<sup>14</sup> note that a naphthene may be transformed into an aromatic hydrocarbon by the action of oxygen at  $140^\circ \text{C}$ . and suggest that a similar change may be accomplished underground by some such agent as manganese dioxide.

Oxygen is usually not determined in the analysis of oil and comparatively little is known of the structure of the oxygen compounds. However, naphthenic acids, which are the oxygen derivatives of the naphthenes, having the general formula  $\text{C}_n\text{H}_{2n} - 2\text{O}_2$ , are known to exist in California oils. According to Ostrejko,<sup>15</sup> if Russian oil, similar to the Californian, is exposed to the air, especially in sunlight, these acids form at ordinary temperature, with resulting turbidity and darkening of the oil. Hydroxyl derivatives of the nature of phenols have been found in

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<sup>10</sup> S. F. and H. E. Peckham: On the Sulphur Content of Bitumens, *Journal of the Society of Chemical Industry* (1897), **16**, 996-997.

<sup>11</sup> H. Endemann: *Journal of the Society of Chemical Industry* (1897), **16**, 426.

<sup>12</sup> W. P. Jenney: On the Formation of Solid Oxidized Hydrocarbons Resembling Natural Asphalts by the Action of Air on Refined Petroleum, *American Chemist* (1875), **5**, 359.

<sup>13</sup> C. F. Mabery, and J. H. Byerly: The Artificial Production of Asphalt from Petroleum, *American Chemical Journal* (1896), **18**, 141.

<sup>14</sup> J. Hausmann, and S. Pilat: Studien über die Oxydation der Petrolkohlenwasserstoffe, *Congrès International du Pétrole, Compte rendu, Sess. 3* (1907), 378.

<sup>15</sup> R. A. Ostrejko: Influence of Sunlight and Air on Petroleum Products (Abstract), *Journal of the Society of Chemical Industry* (1896), **26**, 345 and 645.

California and other oils,<sup>16</sup> and formic and oxalic acids are reported in petroleum from the Grosny district, Russia, the quantity increasing with the gravity of the oil.<sup>17</sup>

The ease with which a petroleum takes up oxygen naturally varies according to the prevailing series of the hydrocarbons in it. For example, the terpenes, which have the general formula  $C_nH_{2n-4}$ , and which are important constituents of some California petroleum, have a well-known tendency to oxidize and to polymerize, and doubtless contribute to the viscosity and resinification of the heavy oils.<sup>18</sup> "However, hydrogen is given up more or less readily by all hydrocarbons in contact with air; even the gasolines soon show color during distillation, and the constituents with higher boiling points more readily become dark and thick. On account of this tendency at high temperatures to lose hydrogen, even the lighter hydrocarbons in varieties of petroleum that contain no asphaltic members are converted during distillation, as ordinarily conducted in the refinery, into the heavier forms that compose the residuums and asphaltic tars. In some crude oils, such for example as the Russian, which is composed largely of the naphthenes, the sudden inflow of air into the hot vapor during distillation in vacuum causes a violent explosion."<sup>19</sup>

#### *Relation of Gravity to Sulphur Content of Oil from California Fields*

As shown in the preceding section, there is no question as to the ability of sulphur and oxygen to render petroleum heavier and more asphaltic, although many of the reactions involved are not well understood. If the action of these agents on petroleum in nature has really been widespread, it is reasonable to expect that variation in the gravity of the oil from any one field would be accompanied by variation in the sulphur and oxygen content, as shown by analysis. It is evident, however, that there can be no exact relation, for the particular part of the sulphur or oxygen that contributes chiefly to increasing the gravity of the oil is the part given off and lost as  $H_2S$  or  $H_2O$ ; and the size of this part in relation to that remaining in the oil as sulphur or oxygen compounds is apparently indeterminate. In other words, only the *traces* of the reactive substances can be found by analysis; the portions of those substances that have actually been effective have been lost. Another factor that might militate against an exact relation is the tendency of the sulphur to concentrate in the

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<sup>16</sup> C. F. Mabery: The Composition of American Petroleum, *Journal of the American Chemical Society* (1906), **28**, 428.

<sup>17</sup> N. Schidkoff: Acid Content of Grosny Petroleum and Derivatives (Abstract), *Journal of the Society of Chemical Industry* (1899), **18**, 360.

<sup>18</sup> F. W. Bushong: Chemical Composition of Petroleum, *Kansas Geological Survey* (1908), **9**, 306.

<sup>19</sup> C. F. Mabery: *Economic Geology* (1916), **11**, 518-519.

heavier fractions of the oil, so that if any evaporation had taken place the sulphur content would become abnormally high. Finally, inasmuch as oxygen is not determined in the industrial analysis of petroleum, the data are not available for studying variation in gravity in relation to oxygen content, and as all gravity variation in all oils can not be ascribed to sulphur alone the relation between sulphur and gravity cannot be uniformly exact. In view of these considerations, the relation between the two shown in the accompanying diagrams is surprisingly close.

Figs. 1, 2, and 3 show the relation between the gravity of the oil and the per cent. of sulphur in more than 300 samples from the Coalinga, Midway-Sunset and Santa Maria fields, California. These fields were selected partly because their product exhibits a considerable range in gravity and sulphur content and should therefore afford a fair test, and partly because of the large number of analyses available. Most of the analyses were made by the United States Bureau of Mines, and all of the published analyses of oil from the fields mentioned that have been made by the Bureau of Mines are plotted in the figures.<sup>20</sup> A few other analyses made in various laboratories have also been included; these are distinguished in the diagrams by circles.<sup>21</sup> According to the method of analysis in the Bureau of Mines the specific gravity of the oil is determined by means of the Westphal balance, or in some cases when the sample is very small by means of the Nicoll pycnometer. Sulphur is determined by carefully washing out with distilled water the contents of the Berthelot combustion bomb, after the combustion of the oil, and precipitating with barium chloride the sulphuric acid so dissolved. As the analyses were, of course, not made with the special purpose of showing relationship between gravity and sulphur, it is possible that in some cases these determinations may be slightly inaccurate; but the large number of analyses involved, and the fact that the great majority of them were made according to a uniform method, reduce to a minimum the chance of erroneous deductions.

Fig. 1 represents 171 analyses of oil from the Coalinga field. All of the Coalinga petroleum is low in sulphur, but the relation between sulphur and gravity as shown in the figure is remarkably delicate. The product of the Coalinga field includes two rather distinct varieties of

<sup>20</sup> The analyses are taken from the following reports. Coalinga field: *U. S. Geological Survey, Bulletin 398* (1910), 264-272; *U. S. Bureau of Mines, Bulletin 19* (1912), 20-21; *U. S. Bureau of Mines, Technical Paper 74* (1914), 34-35.

Midway-Sunset field: *U. S. Bureau of Mines, Bulletin 19* (1912), 24-27, and *Technical Paper 74* (1915), 36-37.

Santa-Maria field: *U. S. Bureau of Mines, Technical Paper 74* (1915), 32-33.

<sup>21</sup> These analyses are cited by D. T. Day: *Production of Petroleum, U. S. Geological Survey, Mineral Resources* (1913), 1126-1139.

Analyses of Santa Maria oil are cited in *U. S. Geological Survey, Bulletin 323* (1907), 115-118.

petroleum, which are believed to have had their source in different formations.<sup>22</sup> The great bulk of the field's production apparently originated in an Oligocene formation, consisting largely of the remains of diatoms and other organisms, and has since migrated into the overlying Miocene beds in which it is now found. This oil is black, asphaltic in character, carries 0.30 to 0.80 per cent. of sulphur and generally ranges in gravity between

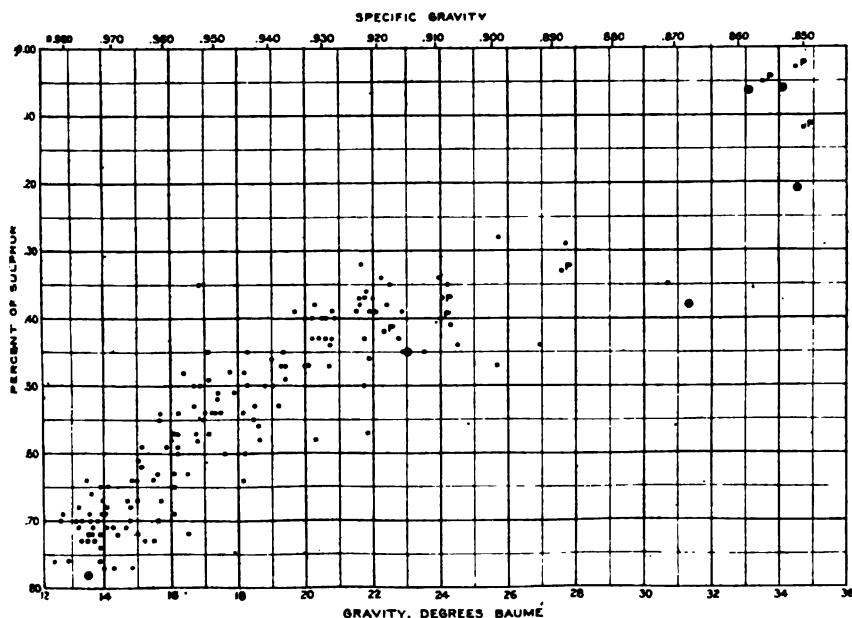


FIG. 1.—RELATION OF GRAVITY TO PER CENT. OF SULPHUR IN 171 SAMPLES OF OIL FROM THE COALINGA FIELD.

Analyses represented by small solid circles made by U. S. Bureau of Mines, the others by various chemists. In analyses marked P paraffin wax was reported.

13° and 30° Bé.; it is represented in Fig. 1 by the great majority of the analyses plotted. The other type of oil apparently originated in another and older diatomaceous shale formation; most of it still remains in that formation, though some has migrated into the overlying Eocene beds. The oil that is in place is greenish in color, contains some paraffin wax, is very low in sulphur and usually approximates 33° Bé. in gravity; it is represented by a few analyses in the upper right-hand corner of the diagram. The oil that has migrated into the Eocene strata also contains

<sup>22</sup> R. Arnold, and R. Anderson: Geology and Oil Resources of the Coalinga District, California. *U. S. Geological Survey, Bulletin 398* (1910), 182-189.

R. Anderson, and R. W. Pack: Geology and Oil Resources of the West Border of the San Joaquin Valley, North of Coalinga, *U. S. Geological Survey, Bulletin 603* (1915), 122-162.

paraffin wax, but is much darker in color and distinctly heavier than what is thought to be the same oil in place. The marked difference between the two main types of oil is usually ascribed to difference in age and probably in the character of the organic remains from which they were derived. This view is doubtless partly correct, but it is probable that some of the difference is due to the fact that the light oil is in place, whereas the heavy has migrated into upper formations and in the course of its movement may well have encountered alterative agents.

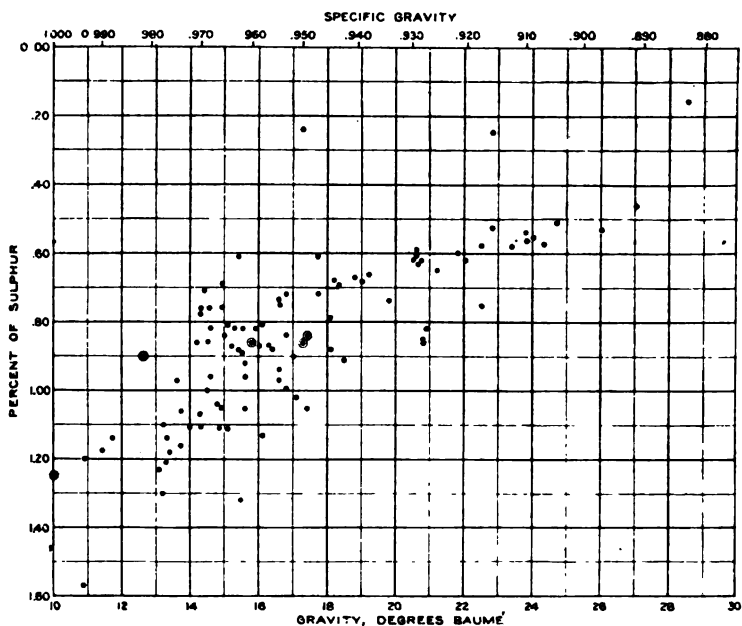


FIG. 2.—RELATION OF GRAVITY TO PER CENT. OF SULPHUR IN 100 SAMPLES OF OIL FROM THE MIDWAY-SUNSET FIELD.

Analyses represented by small solid circles made by U. S. Bureau of Mines, the others by various chemists.

Fig. 2 represents a hundred analyses of oil from the Midway-Sunset field. They exhibit nearly as close a relation between gravity and sulphur as the Coalinga oils, but it is noteworthy that the average per cent. of sulphur is considerably higher. In other words, a typical Coalinga oil carrying a given per cent. of sulphur is considerably heavier than a typical Midway-Sunset oil carrying the same per cent. All of the oil from the Midway-Sunset field is of the same general type, being dark brown to black in color and free from paraffin. It is believed to have originated in diatomaceous shale beds of Miocene age and to have migrated into suitable reservoirs in the overlying formations.<sup>23</sup>

<sup>23</sup> R. Arnold, and H. R. Johnson: The McKittrick-Sunset Oil Region, California, U. S. Geological Survey, Bulletin 406 (1910).



Fig. 3 shows 47 analyses of oil from the Santa Maria field. The relation between sulphur and gravity in these oils appears to be less close than in those already discussed, though this may be due in part to the much smaller number of analyses available. The Santa Maria oils are decidedly higher in sulphur than the foregoing, even the lightest oils containing a larger per cent. of sulphur than the heaviest oils in the Coalinga field. The Santa Maria oil is believed to have originated in a diatoma-

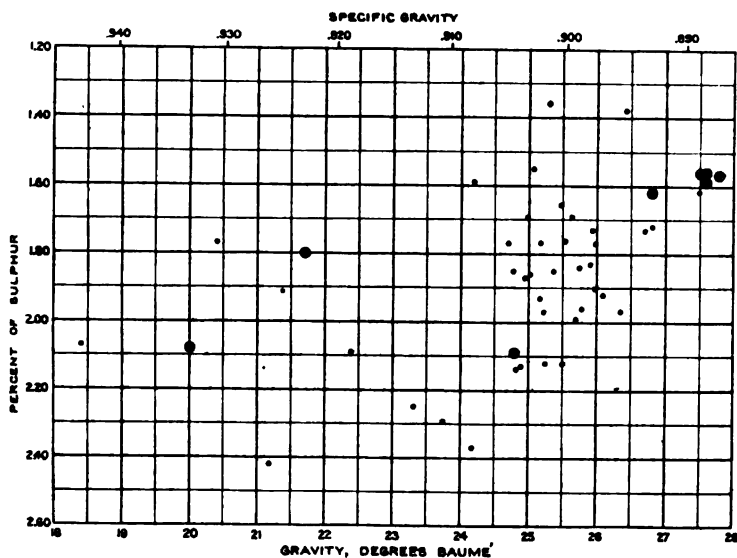


FIG. 3.—RELATION OF GRAVITY TO PER CENT. OF SULPHUR IN 47 SAMPLES OF OIL FROM THE SANTA MARIA FIELD.

These include all analyses published by the U. S. Geological Survey and Bureau of Mines except two, one of which has a gravity of 34.6° B. and 0.60 per cent. of sulphur, and the other a gravity of 15.4° B. and 3.75 per cent. of sulphur.

ceous shale formation.<sup>24</sup> Some of it occurs in fractured zones of flinty shale in the lower part of the diatomaceous beds and some of it has apparently migrated into sand lenses in the underlying formation.

It will be noted that in all three fields there are a few samples of oil in which the sulphur-gravity relation departs more or less widely from the field average. It is not unreasonable to suppose that these departures represent variation in the unknown factor, viz., the oxygen content. Heavy oils abnormally low in sulphur may owe their gravity to excessive

<sup>24</sup> R. Arnold, and R. Anderson: *Geology and Oil Resources of the Santa Maria Oil District, California*, U. S. Geological Survey, *Bulletin* 322 (1907).

oxidation, and light oils that seem abnormally high in sulphur may be free from oxygen and affected only by reaction with sulphur. The great majority, which fall along a regular curve, have probably been affected by both oxygen and sulphur. All of the Coalinga and Midway-Sunset oils that depart most widely from the field average seem to occur in the fairly shallow territory within a mile or so of the outcrop of the sand; but whether this has any special significance is not clear.

### *Possible Sources of Sulphur and Oxygen*

That sulphur and oxygen may be important agents in determining the character of petroleum is evident from the wide distribution of these common elements in available form. A few of the possible sources will be briefly considered.

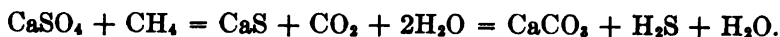
Under ordinary conditions oxygen is probably the less important, though as it is usually not determined in the analysis of oil the extent of its action in nature is conjectural. However, all meteoric waters carry oxygen in solution and it is probable that these waters in places percolate down to the oil measures. Moreover, as Hausmann and Pilat have suggested, oxygen may be contributed locally by oxidizing agents such as manganese dioxide.<sup>25</sup> The most striking evidence of the action of oxygen on California oil is of course seen at the outcrop of the oil sands, where the oil has oozed out on the surface, and, through the evaporation of the lighter constituents and the oxidation of the heavier, has been transformed into asphalt.

There are several possible sources of sulphur, though in the California fields sulphide waters are probably the most important. The surface and shallow groundwaters in the California fields carry large amounts of sodium, calcium and magnesium sulphate, and outside of the oil fields the deeper waters are also strongly sulphatic in character. The waters in and near the oil measures, however, are almost or quite sulphate-free, and are usually solutions of carbonates and chlorides.<sup>26</sup> Between the sulphate surface waters and the sulphate-free waters associated with the oil, every gradation may be found; and near the horizon at which sulphate begins to decrease and carbonate to increase the waters usually contain hydrogen sulphide (see Fig. 4). As sulphate is abundant in the shallower waters everywhere along the California coast ranges, whereas sulphide is found only in the oil fields and near the oil and gas, it is reasonable to suppose that the sulphide has been formed through the reduction

<sup>25</sup> In their geochemical studies of the action of silver solutions on ore-forming minerals Palmer and Bastin point out that certain solutions may act as potent oxidizing agents below the ground water level. *Trans.* (1913), 45, 224.

<sup>26</sup> G. S. Rogers: The Chemical Relations of Oil-field Waters in San Joaquin Valley, California, *U. S. Geological Survey, Bulletin* 653 (1917).

of sulphate by the hydrocarbons. The reaction supposed to be involved is usually written:



Although the field evidence in favor of some such reaction is strong it must be admitted that it has apparently never been experimentally proved.<sup>27</sup> In any event the reaction as written can be considered only as a condensed representation of the type of change that takes place, the intermediate stages in the decomposition of the hydrocarbons on the one hand and of the sulphate on the other being as yet unknown.

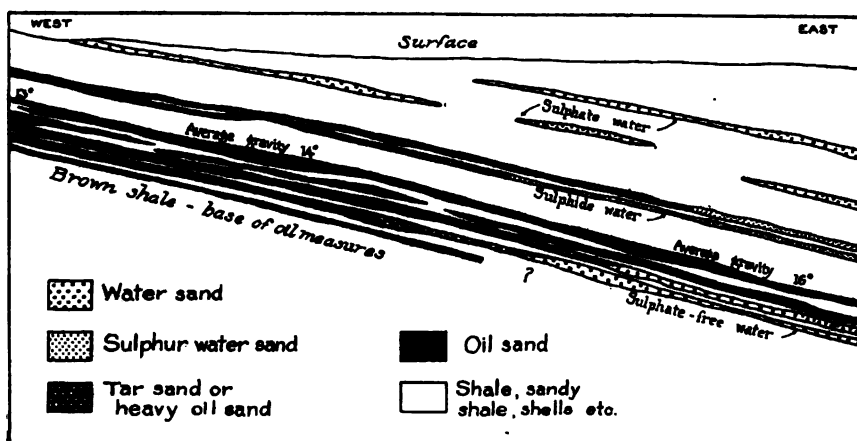


FIG. 4.—GENERALIZED SECTION THROUGH WESTERN PART OF COALINGA FIELD, SHOWING REPLACEMENT OF OIL BY WATER DOWN THE DIP, IN THE TAR SAND ZONE AND IN THE LOWER PART OF THE PRODUCING OIL ZONE. THE OIL IN GENERAL BECOMES HEAVIER TOWARD THE OUTCROP, BUT IN THE SANDS CARRYING BOTH WATER AND OIL A BELT OF HEAVY OR TARRY OIL IS REPORTED TO SEPARATE THE TWO.

In some regions gypsum may be disseminated through the strata near the oil measures and if taken into solution and carried to the oil may be reduced to sulphide. This is, of course, essentially the same as the reduction of sulphate surface waters. Gypsum in the anhydrous condition, however, is a very stable compound, and even with an active reducing agent such as carbon monoxide, a temperature of about 700° C. is required for its reduction.<sup>28</sup>

Whether the reduction of sulphate by hydrocarbons takes place or

<sup>27</sup> Suggestive experiments are however described by K. Kharitschkoff: *The Waters in Petroleum Wells*, *Petroleum Review* (1913), 29, 368.

<sup>28</sup> H. O. Hofman, and W. Mostowitsch: *The Reduction of Calcium Sulphate by Carbon Monoxide and Carbon, and the Oxidation of Calcium Sulphide*, *Trans.* (1910), 41, 763.

not, it is certain that many of the oil-field waters carry hydrogen sulphide or alkaline sulphide in amounts ranging up to more than 300 parts per million. The tendency of alkaline sulphide to become free  $H_2S$ , and the tendency of this gas to oxidize to sulphur, are well known. In this connection the following personal communication from Clifford Richardson is of interest:

"Some years ago I collected in a sealed tube 200 or 300 c.c. of a natural gas in Trinidad which contained hydrogen sulphide. This was allowed to stand for about 10 years without observation, but at the end of that time it was found that the sulphur of the  $H_2S$  was deposited on the walls of the tube in colorless crystals."

The oxidation of hydrogen sulphide proceeds even under very feebly oxidizing conditions, as on the floor of the ocean, and it doubtless takes place even in deeply buried strata.<sup>29</sup> In the light of certain other corroborative evidence,<sup>30</sup> it seems probable that considerable amounts of hydrogen sulphide are oxidized to sulphur, which is precipitated. As the strata directly above the oil measures have not been tested for sulphur, this supposition can not be definitely proved, but it is significant that small deposits of disseminated sulphur are not uncommon along the western edges of the Coalinga and Midway-Sunset fields. Moreover, a commercial deposit of sulphur has been found near the southern end of the Sunset field, in the same formation that contains the oil measures in the field nearby. An interesting feature of this sulphur, to which the writer's attention was first directed by E. A. Starke of the Standard Oil Co., is its intimate mixture with hydrocarbon material, which seems to constitute 20 per cent. or more of the amorphous substance.

It may be added that the waters associated with the oil in many regions are known to be free from sulphate.<sup>31</sup> In many fields there are strong chloride waters which doubtless represent the sea water entrapped in the sediments when they were laid down, and which therefore never contained a very large concentration of sulphate; but in some fields the low chloride and the high carbonate indicate that the waters are in part altered meteoric waters from which considerable quantities of sulphate have been removed. In certain Tertiary and Cretaceous fields in which the normal surface waters are strongly sulphate in character the reduction

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<sup>29</sup> J. Y. Buchanan: On the Occurrence of Sulphur in Marine Muds and Nodules and Its Bearing on Their Mode of Formation, *Royal Society of Edinburgh Proceedings*, (1890-91), 18, 17.

F. W. Clarke: The Data of Geochemistry, third edition. *U. S. Geological Survey, Bulletin* 616 (1916), 514.

<sup>30</sup> G. S. Rogers: *Op. cit.*, 100.

<sup>31</sup> C. Engler, and H. Höfer: *Das Erdöl*, Band 2 (1909), 28.

A. Potilitzin: *Deutsche chemische Gesellschaft, Berichte* (1882), 15, 3099.

E. H. Pascoe: *Memoirs Geological Survey India* (1912), 40, pt. 1, 221.

K. V. Kharitschkoff: *Chemisches centralblatt* (1907), 295.

of the sulphate by the stages outlined above may afford abundant supplies of sulphur to react with the oil. Hence, as meteoric waters carry oxygen and also salts that eventually yield sulphur, it is probable that in many regions waters are the chief agents in the alteration of the oil. The apparent increase in the gravity of oil that has been associated with certain types of water is recognized by many practical oil men (see Fig. 4).

As pyrite is said to react with and yield sulphur to petroleum it is probable that in some localities the action of both pyrite and its less stable isomer, marcasite, have been important. These minerals have been encountered in many wells in the California fields. They probably formed in part during the deposition of the sediments, through the reducing action of organic matter on iron sulphate solutions; but they may also have originated later through the direct action of hydrogen sulphide on chalybeate waters. J. J. Hern, a California oil man of wide experience, told the writer that in wells in which large quantities of iron sulphide are found the oil below is likely to be abnormally warm. If this observation is well founded, it may be significant as indicating chemical reaction between the sulphide and the oil.

In some regions sulphur is doubtless derived from still other sources. Hydrogen sulphide and sulphur dioxide are common components of volcanic emanations, and the former is found in many thermal springs that are supposed to represent the last stages of igneous activity. Much of the Mexican oil, which is heavy, asphaltic, and high in sulphur, is found near igneous intrusions and may well have been affected by the sulphurous gases that doubtless accompanied them. The oil in the salt domes of the Gulf Coast is somewhat similar in character and has evidently been altered by sulphur, but the origin of the sulphur in this case is related to that of the salt domes themselves, and has never been satisfactorily explained. Again, it has long been known that certain bacteria have the property of generating hydrogen sulphide through the reduction of sulphate solutions. Some of these bacteria are anaerobic, being able to exist in the absence of air, and their action has been repeatedly observed in ocean water;<sup>22</sup> but whether they can exist and function in deeply

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<sup>22</sup> A. Van Delden: Beitrag zur Kenntnis der Sulfatreduktion durch Bakterien, *Centralblatt Bakteriologie*, Band 11, Abt. 2 (1903), 92-94, 113-119.

A. Lebedinzeff: Vorläufige Mitteilung über den chemischen Untersuchungen des Schwarzen und Asowischen Meeres in Sommer, 1891, *Soc. Naturalistes à Odessa Trav.* (1891), 16, fasc. 2, 149. Abstract in *Royal Geographical Society Proceedings*, New ser. (1892), 14, 461.

N. Zelinsky: Sulphydric Fermentation in the Black Sea, *Russian Chemical Society Journal* (1894), 25, 298-303. Abstract in *Chemical Society Journal* (1894), 66, pt. 2, 200.

N. Andrussoff: Physical Exploration in the Black Sea, *Royal Geographical Society Geographical Journal* (1893), 1, 49.

buried strata is open to question. Finally, there are oils, like those of the Appalachian fields, that have apparently never been subjected to the action of sulphur; and others, like the Trenton limestone oil of Ohio, that are generally supposed to owe their sulphur to the character of the organic remains from which they were formed.

*Sulphur and Gravity in Relation to the Migration of the Oil*

In the preceding pages the writer has endeavored to show that the alteration of petroleum by sulphur and oxygen in nature is chemically and geologically possible, and the specific field evidence tending to show that such alteration has actually taken place may now be considered. Arnold and Anderson, in their exhaustive report on the Coalinga district,<sup>33</sup> make the significant statement that two of the important factors influencing the gravity of the petroleum are the extent to which it is, or has been, associated with water, and the extent to which it has migrated, either upward, or along a particular bed, or downward. In the present writer's opinion, the increase in gravity incident to association with water and to migration is due not to the action of  $H_2O$  as such nor to mere movement, but rather to the salts and gases carried by the  $H_2O$  and to the reagents that the oil encountered in the course of its movement.

It is evident that in any one field the action of both oxygen and sulphur on the oil will be more or less localized, since ordinarily the quantity of these substances available is insignificant as compared with that of the petroleum. The free chemical energy of reacting substances and the time of contact are potent factors in chemical change, and should be recognized in accounting for variations in the properties of oil that has migrated. An oil that is the first to traverse a given course comes into contact with reacting substances at their highest potential, and therefore becomes changed more radically than does the oil that follows it at the same rate. As soon as these reacting substances have become exhausted, then oil may pass them unaffected. Again, oil that moves with extreme slowness and remains a very long time in contact with reacting substances may undergo changes just as marked as though it had moved more rapidly in a new channel. In general, therefore, that portion of the oil which migrated first or farthest will be the most altered. Moreover, the oil nearest the surface or nearest the outcrop of the oil-bearing zone may be further altered by fresh supplies of descending sulphate waters. Therefore, the oil around the edges—particularly the upper edges—of the main body should as a rule be the most altered or, in other words, the heaviest and most asphaltic.

This reasoning is well borne out by the variations in the gravity of the oil in the Midway-Sunset and Coalinga fields. In the deeper portion of

<sup>33</sup> U. S. Geological Survey, *Bulletin* 398 (1910), 186.

the Midway-Sunset field, for example, the specific gravity of the oil ranges between  $20^{\circ}$  and  $30^{\circ}$  Bé., but as the outcrop is approached the oil becomes heavier and most of the wells nearest the outcrop produce oil of about  $12^{\circ}$  to  $14^{\circ}$  Bé. gravity. A part of this difference may be due to the escape of the more volatile constituents of the oil in the zone along the outcrop, but variation in gravity several miles away from the outcrop can hardly be explained in this way. As shown in Fig. 4, conditions in the Coalinga field are similar, the average gravity increasing from about  $16^{\circ}$  to about  $13^{\circ}$  Bé. in less than 2 miles. Certain producing oil sands in the lower part of the oil measures become water-bearing down the dip, however, and it is significant that the oil and water in these sands are apparently separated by a belt of tar or abnormally heavy oil. In the western portions of the Coalinga, Midway, and Sunset fields, the producing oil sands are overlain at a distance of several hundred feet by the tar sand zone, which contains sands partly impregnated with heavy, viscous tar. Some of the sands in this zone carry sulphur water and most of the tar sands become water bearing farther down the dip. So far as the writer knows, this very heavy tar has never been analyzed, but the analyses of several samples of oil of  $11^{\circ}$  Bé. gravity show about 1.15 per cent. of sulphur, and it may be presumed that the tar carries at least as much as this. In a general way the tar sand zone marks the farthest limit of migration of the oil. It has been observed, furthermore, that oil which has migrated for some distance into sands that lie in angular unconformity with the main oil zone generally becomes heavier with distance from the main body and finally passes to tar. Finally, the marked difference in character between the light Coalinga oil that has migrated only a short distance, if at all, and the heavy Coalinga oil that is believed to have migrated from one formation to another, may be due in part at least to the alteration of the heavy oil in the course of its movement. Purely on the basis of field evidence, therefore, it would appear that among the important factors influencing the gravity of the oil are the distance that the oil has migrated and the extent to which it is or has been subject to contact with waters, especially meteoric (sulphate) waters.

There seems little doubt, therefore, that the local variations in the character of the oil are in part, at least, due to the action of sulphur and probably also of oxygen. Sulphate waters descending from the surface are to a large extent altered in the zone of tar sands, and the tar itself is thereby rendered still more asphaltic. To some extent, therefore, the tar sands may be conceived as protecting the main body of oil in the sands below. The same process goes on near the outcrop, although how far descending meteoric waters have affected the main body of the oil since it attained its present position is a matter of conjecture. It seems more reasonable to suppose that the oil which migrated first and farthest was

considerably altered by the water that had previously occupied the sands, and that most of its alteration took place before it had come to rest in its present position.

*Bearing of the Reactions Outlined on the Migration and Origin  
of Oil in General*

Although the action of sulphur and oxygen thus seems to have been important in determining the local character of California oils, the writer realizes that conditions in the Californian fields are exceptional. There are few regions in which commercial accumulations of petroleum occur so near the outcrop of the containing sands and in which, therefore, the oil is so likely to be affected by meteoric waters. Carbonate waters similar to those of the California oil fields are described from the Russian, Galician and Burma fields and have doubtless played a similar rôle in the alteration of the petroleum in those areas. In most regions, however, meteoric waters do not appear to have had access to the oil measures, and the oil-field waters are strongly salty. The very fact that the oil itself can accumulate and remain within a small area under high pressure for long periods of time ordinarily implies that the underground circulation is decidedly restricted. Hence, if the beds were originally laid down in salt water it is reasonable to expect that this water under most conditions would be retained with the oil, and other water excluded; and as sea water is relatively low in sulphate its action on the oil would not be very pronounced. It is only when oil is brought into contact with a type of water carrying reactive agents—either through the migration of the oil into a formation saturated with such water, or through the percolation of the water to the oil zone—that the action of water on oil is likely to be localized and therefore apparent.

There is a tendency among petroleum geologists to assume that many oil pools represent the accumulation of the oil originally disseminated over a wide area. The enormous deposits in some small districts are indeed difficult to explain except as due to concentration. If the supposed course of the oil is along one sand, or laterally in one formation of reasonably homogeneous character, the hypothesis may be acceptable; but it should be recognized that if the oil has traversed beds of diverse texture and containing reactive substances it is very likely to have undergone recognizable change. Contact with sulphur and oxygen may result in the loss of hydrogen and the formation of unsaturated and asphaltic compounds; subsequent passage through shale may result in the fractionation of such oil by diffusion, and the removal of the heavy fractions, as suggested by the experiments of Day.<sup>34</sup> Hypotheses involving the migra-

<sup>34</sup> D. T. Day, J. E. Gilpin, and M. P. Cram: The Fractionation of Crude Petroleum by Capillary Diffusion, *U. S. Geological Survey, Bulletin* 365 (1908).

J. E. Gilpin, and O. E. Bransky: The Diffusion of Crude Petroleum through Fullers Earth, *U. S. Geological Survey, Bulletin* 475 (1911).



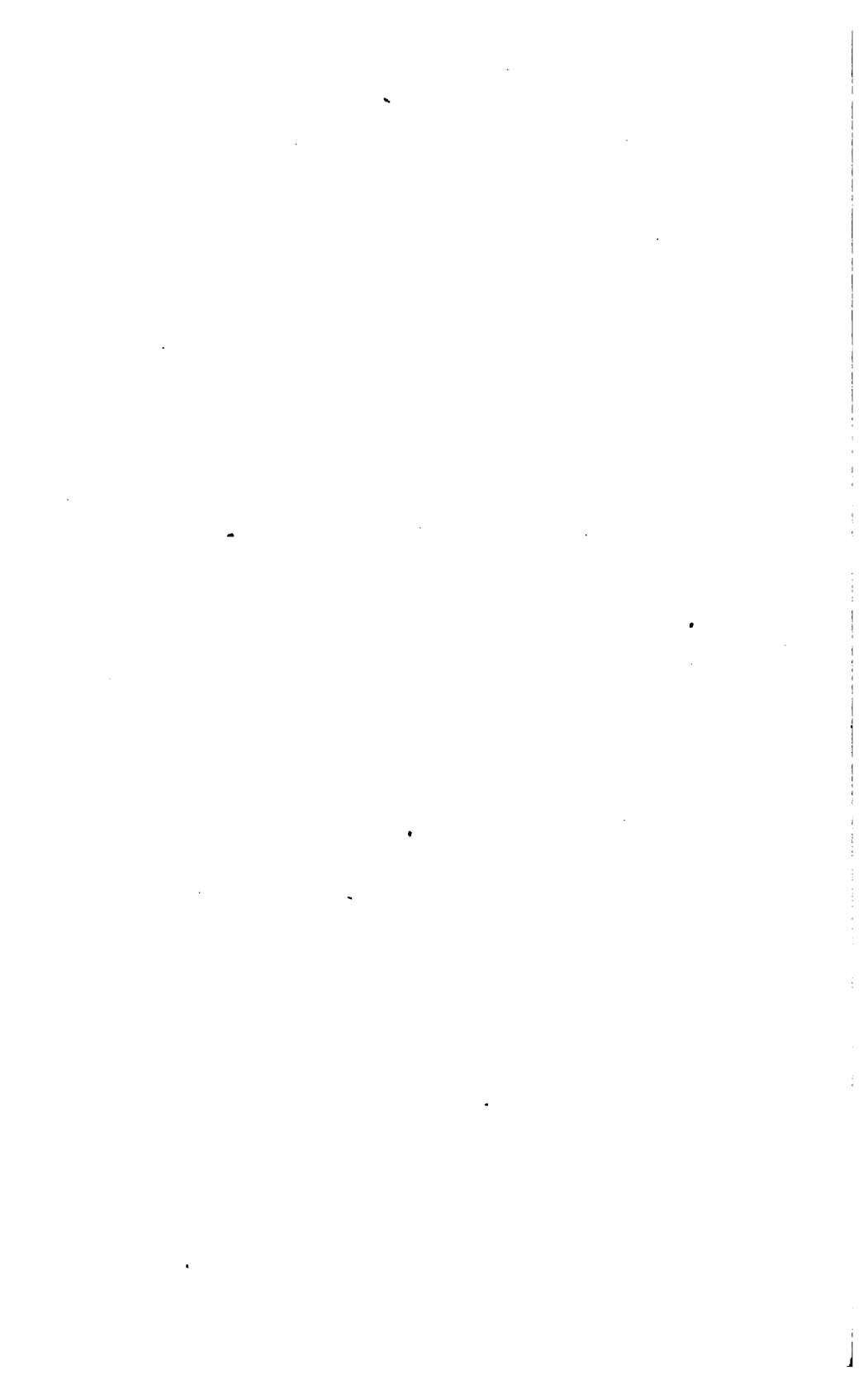
tion of apparently unchanged oil over long distances should be accepted with caution, for petroleum is one of the most complex and delicate of natural substances.

The reactions outlined have a bearing also on theories of the origin of petroleum. Thus, they appear to furnish further evidence against the inorganic hypothesis, for in view of the pronounced changes induced in the California oils by a relatively short migration it is difficult to understand how petroleum could have ascended from the interior of the earth. Some of the Mexican oils, which are unusually heavy and high in sulphur, have been cited, because of their apparent connection with igneous intrusions, as evidence in favor of inorganic origin, but their peculiar character is simply what would be expected in any oil that has been locally exposed to the action of gaseous emanations, such as hydrogen sulphide.

If the organic origin of petroleum is accepted, it is evident that the old idea, that sulphur in petroleum indicates derivation from animal remains, is not necessarily valid, for any oil may take up sulphur from extraneous sources. On the other hand, there is no reason to suppose, since contact with sulphur increases the gravity of oil, that the earlier stages in the evolution of petroleum are represented by the light sulphur-free oils and the older or higher ranks by the heavier oils. David White<sup>35</sup> has shown that the character of petroleum depends largely on its degree of dynamic alteration, the light oils being in general the oldest and most altered and the heavy oils the youngest and least altered. This is indicated by the fact that the gravity of oil, regionally considered, varies inversely with the degree of metamorphism of the coal in neighboring strata. It is generally recognized, furthermore, that the character of the original organic material has a bearing on the composition of the oil derived from it, though the precise relation has not yet been ascertained. Day's work has indicated that another important factor is probably the character of the rock—whether coarse or fine, loose or indurated—in which the petroleum is found or through which it has moved. These are generally the broad and fundamental factors; the action of sulphur, oxygen or other agents is usually local and in a measure incidental, though it must none the less be taken into account.

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<sup>35</sup> David White: Some Relations in Origin between Coal and Petroleum. *Washington Academy of Science Journal* (March 19, 1915), 5, No. 6, 189-212.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## **The Metallurgy of Lead Ores in the Lower Mississippi Valley**

BY HERMAN GARLICH, E. M., ST. LOUIS, MO.

(St. Louis Meeting, October, 1917)

THE development of the extensive Southeast Missouri deposits greatly preceded that of the Iowa and Wisconsin deposits. It began about 1720 at Mine La Motte and other localities, and has continued uninterruptedly to the present time. It is estimated that this district produced about 184,000 tons of lead in the year 1916, having a value of approximately \$25,000,000. This is a new high record.

The Southwest Missouri orebodies were hardly known before 1845 and were not extensively developed until 1870. Lead production in Southwest Missouri for 1916 was 38,788 tons. Oklahoma produced 14,399 tons and Kansas 2,345 tons, making a total for Southwest Missouri, and the adjoining districts in Oklahoma and Kansas, of 55,532 tons of lead.

The first metallurgists in Missouri were the Indian and the hunter. The early settler learned to procure his bullets either by melting ore in his camp fire or by throwing pieces of galena on an old stump and depending on the usual roasting and reaction method to obtain the metal. A little more refined method was to arrange two flat stones in the form of a V. Wood and galena were placed in the furnace and ignited. As heat developed, some of the metal was extracted and molded into bullets.

The next step in utilizing the lead ores was the log hearth. All lead in Missouri was smelted in this crude affair before 1820. It was built on sloping ground, and consisted of a hearth of stone, surrounded on the front and two sides by a stone wall. The wall was 7 ft. high in front. The top and rear end were left open and in front an arch or opening was made, forming the eye of the furnace. In front of this a pit was dug in the ground, to receive the molten metal. Large logs were rolled in at the back and made to rest upon ledges formed inside, to raise them from the hearth and to give a draft. These logs filled the entire width of the furnace. Small split logs were then set up around the two sides and the front, and the ore was piled on until the furnace was full. Finally, the mass was covered with logs and fuel until the ore was completely surrounded. A gentle heat was started, which was raised very gradually. After 12 hr. the heat was increased and continued for 12 hr. more, 24 hr. being required for each smelting. The furnace treated about 5,000 lb. per 24 hr. and the ore yielded about 50 per cent. of its metallic lead. A considerable quantity of the ore was not desulphurized, and fell between

the logs into the ashes, forming a kind of slag, which was called "lead ashes." This was rich in lead and was frequently treated in a furnace of a peculiar construction, called an "ash furnace." This ash furnace was introduced from Virginia and was a crude reverberatory, built of limestone, which lasted 15 to 20 days on continuous work. As the ashes or residues were more or less oxidized, the charge was immediately reduced at a moderately high temperature. Silica as chert or sand was mixed with the charge. In about 2 hr. the furnace was ready to tap. The slag was tapped first, then the lead on the opposite side of the furnace. The total extraction from the two operations was about 75 per cent. of the lead contained in the ore.

The next step in the progress of metallurgical operations in Missouri was the reverberatory furnace patterned after practices in Carinthia, Silesia and England. At present this type of furnace is no longer in operation. The Desloge Consolidated Lead Co. operated the Flintshire furnace on a small scale until recently.

Ingalls<sup>1</sup> gives the daily capacity as 5 tons of dry concentrates, the cost at \$7 per ton of concentrate and the extraction at 91 per cent. To this cost should be added the overhead expense which, in a moderate-sized plant, might increase the cost to \$8 per ton.

The reverberatory requires pure ores and on this class of ores gave fair results, considering that no attempt was made to recover the fumes.

The Backwoods hearth was another type utilized for the reduction of Missouri ores and was extensively used in both the Southeast and Southwest districts. The early ones were built entirely of stone and the blast was supplied by bellows.

An improvement was the water-backed hearth, which permitted of continuous work. The Jumbo hearth of Joplin was an enlarged furnace of this character. Before the recovery of the lead fume, these furnaces extracted from pure ores about 70 per cent. of lead contained, not considering the values in the gray slag and fume.

It was not until E. O. Bartlett, in the 70's, installed the bag house for recovering the fume that the hearth could be considered an economic furnace.

On account of the purity of the Southwest galena, the Lone Elm Smelting Co. (now the Eagle-Picher Lead Co.) volatilized all lead possible in short shaft furnaces (slag-eyes), producing a pigment which they called sublimed white lead, being a so-called basic lead sulphate consisting of  $\text{PbSO}_4$  with varying percentages of  $\text{PbO}$  and  $\text{ZnO}$ , the zinc being derived from the Joplin ores, which carry from 2 to 3 per cent. zinc in the high-grade concentrates.

The Cupola process was the next one in line, and differed from the methods just described in that it divided the operations into two distinct steps, each performed in separate furnaces. The concentrates were

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<sup>1</sup> *Engineering & Mining Journal*, Dec. 16, 1905.

roasted in the usual reverberatory 55 to 60 ft. long at gradually increasing temperatures. By later additions of sand, the roasted concentrates were sintered and partly slagged. The usual amount of sulphur left in the sinter was 5 to 6 per cent. To reduce the sulphur further meant increased temperature to decompose the lead sulphate with resulting heavy losses in volatilized lead. The round (Pilz) blast furnace for reducing the sintered product was later replaced by the rectangular (Raschette) furnace. This method allowed a large production of lead, but the total extraction was not over 90 per cent., due to losses in sintering, smelting and retreatment of mattes.

The reverberatory and hearth methods require pure ores. On this class of ores, these processes give fair results when care is taken in smelting. The yield ranges from 80 to 90 per cent. of the lead in the ores, allowing no recovery in fume. The blast furnace preceded by roasting in reverberatory is adapted to ores that are less rich in lead, and contain impurities which would cause severe losses if treated by any other process. Also, when there is a large amount of ore to be treated with scarcity of labor, the blast-furnace process is the more economical.

I have passed lightly over these old practices, all of which, at least in part, are now abandoned. I obtained much of my information from *Missouri Geological Survey*, Volume VI; *Illinois Geological Survey, Bulletin* 21; and Pulsifer: *Notes for a History of Lead*.

*Analyses of Ores as Mined in Southeast Missouri, and of Various Grades of Resulting Concentrates*

|                                      | Ag.<br>Oz. | Cu,<br>Per<br>Cent. | Pb,<br>Per<br>Cent. | SiO <sub>2</sub> ,<br>Per<br>Cent. | Fe,<br>Per<br>Cent. | Al <sub>2</sub> O <sub>3</sub> ,<br>Per<br>Cent. | CaO,<br>Per<br>Cent. | MgO,<br>Per<br>Cent. | Zn,<br>Per<br>Cent. | S,<br>Per<br>Cent. | Ni and<br>Co, Per<br>Cent. |
|--------------------------------------|------------|---------------------|---------------------|------------------------------------|---------------------|--|----------------------|----------------------|---------------------|--------------------|----------------------------|
| Ore.....                             | 0.12       | 0.06                | 5.7                 | 5.0                                | 4.1                 | 4.9  | 25.5                 | 14.2                 | 0.8                 | 2.0                |                            |
| High-grade<br>concentrates ...       | 0.7        | 0.13                | 73.2                | 1.0                                | 3.5                 | ...  | 2.6                  | 0.8                  | 0.4                 | 15.0               | 0.05                       |
| Medium<br>concentrates ...           | 1.3        | 0.12                | 68.6                | 1.4                                | 4.6                 | ...  | 3.1                  | 1.4                  | 0.8                 | 15.5               | 0.06                       |
| Low-grade<br>concentrates ...        | 1.0        | 0.30                | 65.8                | 0.5                                | 3.1                 | 0.5  | 4.3                  | 2.8                  | 1.7                 | 13.7               |                            |
| Flotation slimes ...                 | ...        | 0.50                | 45.0                | 9.6                                | 4.4                 | 3.1  | 7.5                  | 4.2                  | 4.0                 | 12.3               |                            |
| Flotation slimes,<br>High-grade .... | 3.7        | 0.05                | 57.8                | 6.0                                | 2.7                 | ...  | 2.2                  | 1.4                  | 9.4                 | 15.5               |                            |
| Joplin con-<br>centrates .....       | ....       | ....                | 80.2                | 1.1                                | 1.0                 | ...  | 0.4                  | ...                  | 1.7                 | 13.3               |                            |

Analyses of the ores as mined in Southeast Missouri—which are about the average of the district—also analyses of various grades of resulting concentrates, show that the metallurgy is simple, the removal of the sulphur and the slagging of the dolomite in the blast furnace being of prime importance.

There is a little nickel and cobalt in the Southeast Missouri ores, more especially in those found at Mine La Motte and the North American mine near Fredericktown. The nickel and cobalt concentrates with the copper in the mattes, but these metals are eventually lost in the refining of the copper.

James W. Neill<sup>2</sup> made a number of successful blast-furnace runs in the 80's at Mine La Motte with nickel and cobalt containing copper and lead, producing a bullion, a matte low in nickel and cobalt, and a speiss containing a large percentage of nickel and cobalt (23 to 24 per cent.), which was shipped elsewhere to be refined. He obtained the necessary arsenic from Western argentiferous speisses, recovering the silver and gold therein. Later the North American Lead Co. built a plant to separate nickel, cobalt and copper by the Hybinette process. This process produced mattes containing the three metals, which were then separated by a wet method. The plant has been closed for many years, but is now controlled by the Missouri Cobalt Co. and is being remodelled for another campaign.

The big improvements which have revolutionized the metallurgy of the Mississippi Valley ores in the last 10 or 12 years are:

1. The introduction of Huntington-Heberlein pot roasting preceded by a preliminary roast in a mechanical furnace (Godfrey), subsequent smelting in the blast furnace and filtering all fumes through cotton or woolen bags.

2. The introduction of the Dwight-Lloyd sintering process which eliminated preliminary roasting for the concentrates, but not for the mattes, with subsequent smelting in blast furnace with bag-house attachment.

The Godfrey preroasting furnaces were abandoned in the Huntington-Heberlein process, and the Wedge roaster was adopted at Herculanum for the preroasting of mattes. Later it became apparent that elimination of sulphur was still unsatisfactory, and that there was an undesirable, but necessary, dilution of the concentrates from 70 and 65 per cent. to 40 and 45 per cent. in lead to render the Dwight-Lloyd machine applicable with a single roast.

To avoid these objections, the double roast is being installed by both the Herculanum plant of the St. Joseph Lead Co. in Missouri and the Federal plant of the Federal Lead Co. in Illinois. A rapid preliminary roast is given on a separate line of Dwight-Lloyd machines, and this preroasted material is then turned over to another set of Dwight-Lloyd machines (Herculanum), or to Huntington-Heberlein pots (Federal) to reduce the sulphur of the finished product to about 2 to 2½ per cent. and with its lead contents increased to 50 or 55 per cent.

It is expected that the double roast will effect the following improvements over present methods:

1. To avoid the large production of mattes which lock up lead and require expensive retreatment for its recovery.

2. To avoid diluting a 65 to 70 per cent. lead concentrate to 40 to 45 per cent., which means large additions of barren fluxes.

3. To utilize to the fullest extent the fluxing values in the ores. This

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<sup>2</sup> *Trans.* (1884-85), 13, 634.

means that no limestone should be added to blast-furnace charge, while some silica and iron will be required. The resulting blast-furnace slags will contain from 6 to 7 per cent.  $MgO$  with a corresponding increase in blast-furnace capacity for original concentrates.

4. To recover a part of the copper in the concentrates, which metal is at present mostly oxidized and lost in the large amount of slags made.

The other important improvement in the metallurgy of Mississippi Valley lead ores was the mechanically rabbled hearth, known as the "St. Louis hearth." This machine was developed at the Collinsville Plant of the St. Louis Smelting & Refining Co., and has been fully described by W. E. Newnam.<sup>3</sup>

The St. Louis hearth is applicable only to concentrates assaying 68 per cent. or better in lead, and therein differs from the Huntington-Heberlein and Dwight-Lloyd methods which treat concentrates of varying grades.

When concentrates assay over 70 per cent., it is considered that the hearth shows better recovery in lead, and is the most economical method, although the double roast, when perfected, promises to be equally as cheap as to cost, with the advantage of being able to treat all grades of concentrates.

The peculiar advantages of the mechanical hearth for the higher grade concentrates are:

1. It treats the concentrates undiluted with any flux.
2. It is a very efficient desulphurizer, expelling 95 per cent. of the sulphur.
3. It retreats all the dust and fume produced, leaving the gray slag as the only product to be sent to the blast furnace, with a large reduction in slag and matte produced.
4. The hearth makes a total extraction of 82 to 84 per cent. of the lead contents of 70 per cent. concentrates. With concentrates assaying 75 to 80 per cent. in lead, 90 to 95 per cent. represents the total extraction, leaving from 5 to 10 per cent. of lead contents in the gray slag. This practically eliminates any retreatment of mattes.

The mechanical rabbling machine is limited to 68 per cent. lead concentrates, and, for this reason, where lower grades of concentrates are received at Collinsville, the Dwight-Lloyd machine must be used as an auxiliary. All mattes produced by smelting the gray slag of the hearths and the Dwight-Lloyd sinter at Collinsville can be easily handled *raw* on the Dwight-Lloyd, so that a Wedge roaster is not needed for pre-roasting.

The St. Joseph Lead Co. is erecting at Herculanum 12 St. Louis hearths to be used in connection with the double-roast process for the treatment of excess of high-grade concentrates.

In the Southeast district, about 75 per cent. of the output of the mills

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<sup>3</sup>The Newnam Hearth. *Trans.* (1916), 54, 485. Lead Mining and Smelting at Galetta, Ont., *Bulletin* No. 124 (April, 1917), 425.

is a high-grade product, applicable to hearth work, the remainder being lower-grade concentrates and flotation slimes. In order to sweeten the mixture for the double roast (*i.e.*, to raise its grade) it is necessary to use a proportion of the high-grade concentrates for this purpose. The balance of the concentrates should be worked by the mechanical hearth to obtain the most economical results both as to cost and recovery.

The lead produced from Missouri ores contains very few impurities, and when subjected to a single liquation process can be used for all purposes except corroding. A typical analysis of Missouri undesilverized lead is as follows: Ag, 0.0080 (2.4 oz. per ton); As, trace; Sb, 0.0030; Bi, trace; Cu, 0.0800; Fe, 0.0015; Zn, trace; Ni and Co, 0.0080; lead by difference, 99.8995.

The Herculaneum and Collinsville plants desilverize part of their lead, recovering about 1 to 2 oz. of silver per ton, and producing an unusually pure lead for the manufacture of white lead and other purposes requiring a very pure lead.

The following analysis of Southeast Missouri refined, indicates the unusual character of Missouri desilverized lead: Ag, 0.0005; As, trace; Sb, 0.0020; Bi, trace; Cu, 0.0002; Fe, 0.0004; Zinc, 0.0004; Lead by difference, 99.9965.

The difference from Western refined is almost entirely due to the bismuth contents (about 0.05 in Western refined) from which the Missouri ores are singularly free.

The refining of the Missouri crude lead for corroding purposes follows the usual practice, except that no softening furnaces are necessary.

At Collinsville the lead from the blast furnace is poured molten into the drossing kettle. After carefully drossing to eliminate copper as much as possible, it is pumped into a desilverizing kettle. The hearth lead is charged as pig directly into the desilverizing kettle, drossed and desilverized. Zinc is added, stirred in and the zinc-silver-lead alloy (crust) is removed. This first crust is reworked in succeeding kettle charges until sufficiently high in silver to be set aside. The resulting retort bullion (lead riches) assays from 500 to 600 oz. Ag per ton, and represents a concentration of the silver contents of 250 to 300 tons of lead into 1 ton of retort bullion. The zinc consumption per ton of refined lead produced is about the same in amount as is consumed in refining argentiferous bullion. Special attention must be given to removing copper as much as possible from the lead to be desilverized.

The quantity of zinc-silver-lead alloy made is insufficient in quantity to warrant the operations of retort and cupels, and heretofore the alloy has been sent to some outside refinery to be separately distilled in retorts, and returns are based on the sampling of the various products obtained, such as retort bullion and dross, spelter and blue powder. The assay results check quite closely. Retorts and cupels are being installed at Collinsville, so as to be able to treat the alloy whenever occasion demands.



### Notes on Flotation—1916

Discussion of the paper of J. M. CALLOW, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 245 to 275.

ARTHUR CROWFOOT, Morenci, Ariz. (communication to the Secretary\*).—In order to set at rest any doubts in regard to the reasons for selecting the Callow pneumatic flotation process for use in the concentrators of the Arizona Copper Co., Ltd., at Clifton and Morenci, I am presenting the following facts gained from the results of test work carried out on the Callow and Cole-Bergman types of flotation machines at Morenci.

Prior to the comparative test of merit between the Callow and Cole-Bergman machines, many months' time and a large amount of money had been expended in the development and testing of the Cole-Bergman machine, and it was necessary to decide quickly what make of machine was best to install.

The Callow pneumatic flotation process was recommended for installation in our concentrators for the following reasons:

The results indicated:

1. That the Callow system made the highest extraction of copper values from the feed treated.
2. That the Callow system was simpler to operate and could be operated at a lower cost.
3. That the Callow system operated with a lower air pressure. (The question of the actual power consumption of the two types of machines was not definitely settled at this time.)

During the official test period the Callow flotation apparatus showed an additional recovery of from 1 to 1.5 lb. of copper per ton more than the Cole-Bergman machine, this difference being maintained by further treatment of the tailing from both machines.

The simplicity of the operation of the Callow flotation system is one of its notable features, two operators per shift being able to handle 90 standard cells and blower equipment.

At the time of the test we were, unfortunately, unable to obtain exact comparative data on power consumption by Callow and Cole-Bergman flotation machines for the reason that the Cole-Bergman machine used a blower which was working at a higher pressure than that for which it was designed, resulting in an unnecessarily high horsepower consump-

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\* Received May, 1917.

tion. It was considered doubtful, however, on account of the wide difference in the power results obtained, that the Cole-Bergman power consumption could be brought down to as low a figure as the Callow power consumption, and it was evident that it could not be brought to a lower figure; this assumption has since been sustained by the operation of the Callow and Cole-Bergman machines in the No. 4 concentrator at Clifton.

On account of the indefinite data on power consumption, this point was not considered in influencing the decision to adopt the Callow machine.

The above facts were determined from test runs made at Morenci, during which the feed to both types of machines originated in the discharge of Hardinge pebble mills treating secondary table middlings.

With the exception that the feed originated from different mill units, it was identical for both machines. In the Callow system, the mill discharge was sent to a Caldecott cone, the spigot discharge of which was returned, via Wilfley tables, to the original mill for regrinding, the overflow of the cone going to the flotation machines.

In the case of the Cole-Bergman machine, the mill discharge was sent through a special form of Pachuca tank to the first compartment of the flotation machine attached to the bottom of which were three classifier cones. The spigot discharges of the classifiers were returned, via Wilfley tables, for regrinding in the original mill as in the case of the Callow machine, the rest of the feed passing through the two remaining compartments of the machine. In both cases the cleaner middling was returned to the system.

The final decision made in favor of the Callow machine was also influenced by the fact that while the Cole-Bergman machine possessed much merit, further development appeared to be required before it could be considered perfected, and for this there was not time, while the Callow machine was already standardized.

The feeding into a flotation machine direct of a Hardinge mill product that has been ground in the mill to pass a 20-mesh screen appeared to me to be an unnecessary and unscientific method of procedure, for the following reasons:

The mill product contains a considerable percentage of material upon which the flotation machine can do no useful work. This material consists of middling grains that contain included grains of mineral, and also free mineral grains too heavy to be lifted by the bubble column of the flotation machine. All of this material must be removed from the flotation machine, and sent: first, to tables so that the heavy grains of free mineral may be removed; second, to a regrinding mill so that the sand may be crushed to a fine size in order to liberate the included mineral grains; and third, back to the flotation machine again. (The use of tables as mentioned above is dependent upon whether the amount of

coarse free mineral grains contained in the coarse sand is sufficient to make a gravity concentration process profitable.)

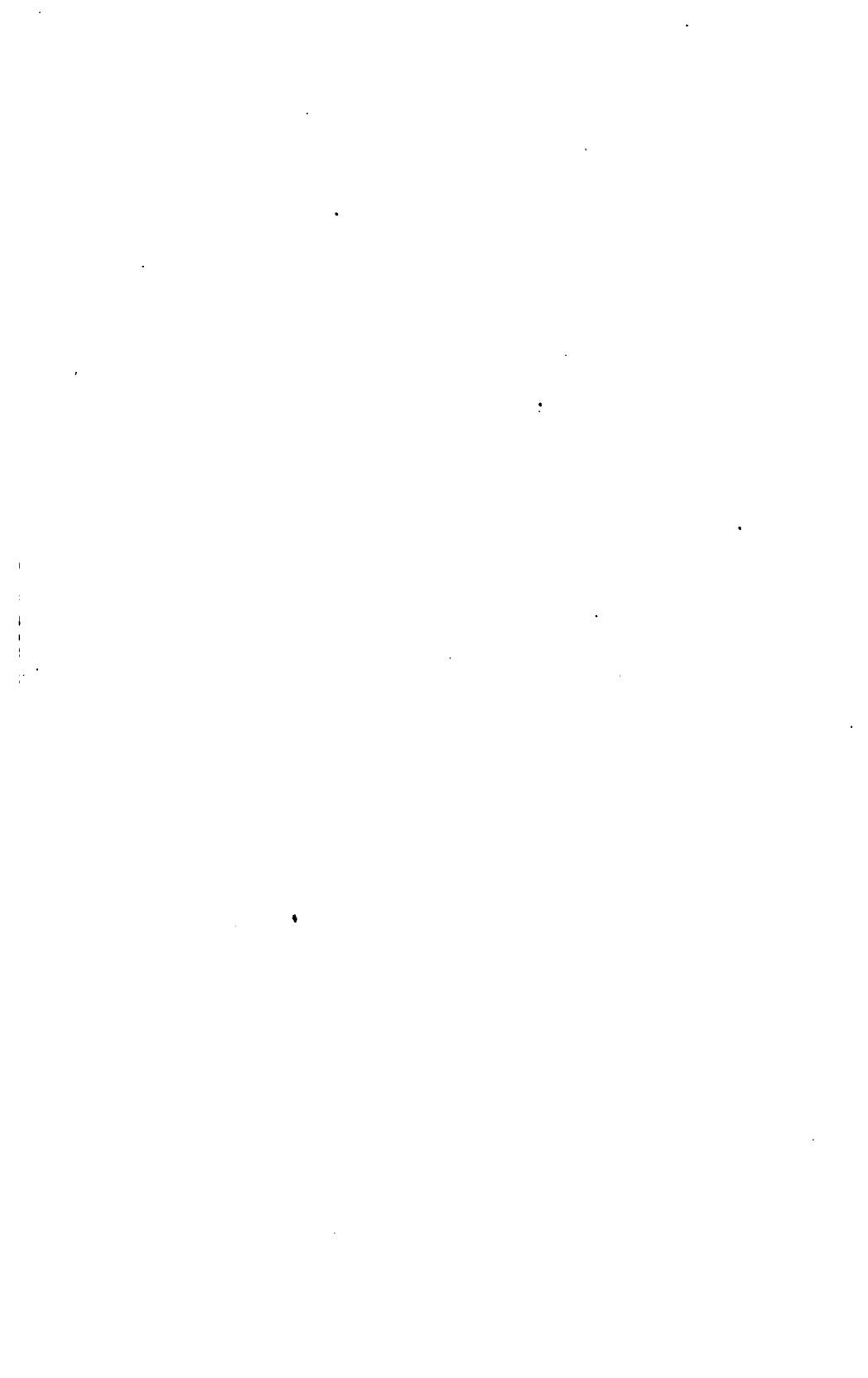
Now the question arises: if these included mineral and heavy free mineral grains are not benefited by being passed through the flotation machine, why should they be sent to the flotation machine before they are properly prepared for treatment? It certainly seems advisable to remove this material before passing the feed on to the flotation machine, by the use of some form of outside classifier, such as a Dorr, Aikens or Allen cone, which can be controlled to advantage, rather than to feed the discharge of the mill directly into the flotation machine and trust to a system of hydraulic classifiers over which we have imperfect control, to remove the sand requiring regrinding and the mineral grains which are too heavy to be lifted by the bubble column.

In our flotation practice we find that we cannot obtain satisfactory results if we send to the machine a feed containing more than 3 to 4 per cent. of material that will rest on a 48-mesh screen; therefore, I can think of no good reason for sending coarser material to the flotation machine, but can see a very good reason for separating the coarse feed from the fine while it is still outside the flotation machine and sending it to the tables where it belongs, recovering the coarse free mineral and then returning the rest to the regrinding mills to be ground to pass 48-mesh.

I look upon the work of the preparation of a flotation feed as the work of a grinding and classifying system which must work independently of the actual flotation operation. The actual flotation work should start with a properly prepared pulp; any attempts so far made to unite the grinding and flotation systems by classifying out the coarse sands in the flotation machine have resulted, to the best of my knowledge, in frequent choke-ups, the use of additional water, more skill and attention required on the part of the operator to keep the system running, and in inferior metallurgical results.

My reasons for not heeding a request to feed the Callow cells with a minus 20-mesh mill product are obvious from the foregoing statements.

The Cole-Bergman machine was removed to our 500-ton mill at Clifton, and operated over a long period of time in parallel and in series with Callow machines. This machine has recently been replaced by a second unit of four Callow cells.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Effect of Anti-Friction Bearings on the Haulage of a Coal Mine

BY P. B. LIEBERMANN,\* NEWARK, N. J.

(St. Louis Meeting, October, 1917)

### 1. *Haulage Tests on Coal-mine Cars*

THE haulage of coal from the face to the tippie is an important enough link in the production of coal to deserve its full share of study and care.

In order to obtain a better understanding of mine haulage conditions, a series of tests and investigations has been made which promises to hasten the improvement of present conditions.

For hauling coal through the mine and on the surface, locomotives are required, the size and power consumption of which depend on the resistance to motion offered by the mine cars. This resistance is principally caused by gravity and bearing friction. The effect of gravity can be reduced to a minimum by a suitable layout of the haulageways; that is, by arranging the grades in favor of the loaded trips, and by fixing the percentage of grade to that value which gives the lowest power consumption for loaded and empty trains. The bearing friction, which is considerably greater than is generally realized, can be easily reduced to a negligible quantity by the installation of a suitable anti-friction bearing that will stand the rough use and abuse of coal-mine service.

The present investigation will deal with flexible roller bearings in comparison with ordinary plain bearings.

A number of tests for determining train resistance have been run during the last 2 years both in the bituminous and anthracite coal regions. For all these tests the dynamometer car described in a previous paper was used. This car is equipped with recording instruments and is entirely automatic in its operation. It is coupled between the locomotive and train, and records the following quantities: drawbar pull in pounds, speed in miles per hour, and a time interval, usually 5 sec.; the paper travel is proportional to the distance covered by the car. Opposite these records the profile of the road is drawn in by hand so that full particulars

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\* Chief Engineer, Hyatt Roller Bearing Co.

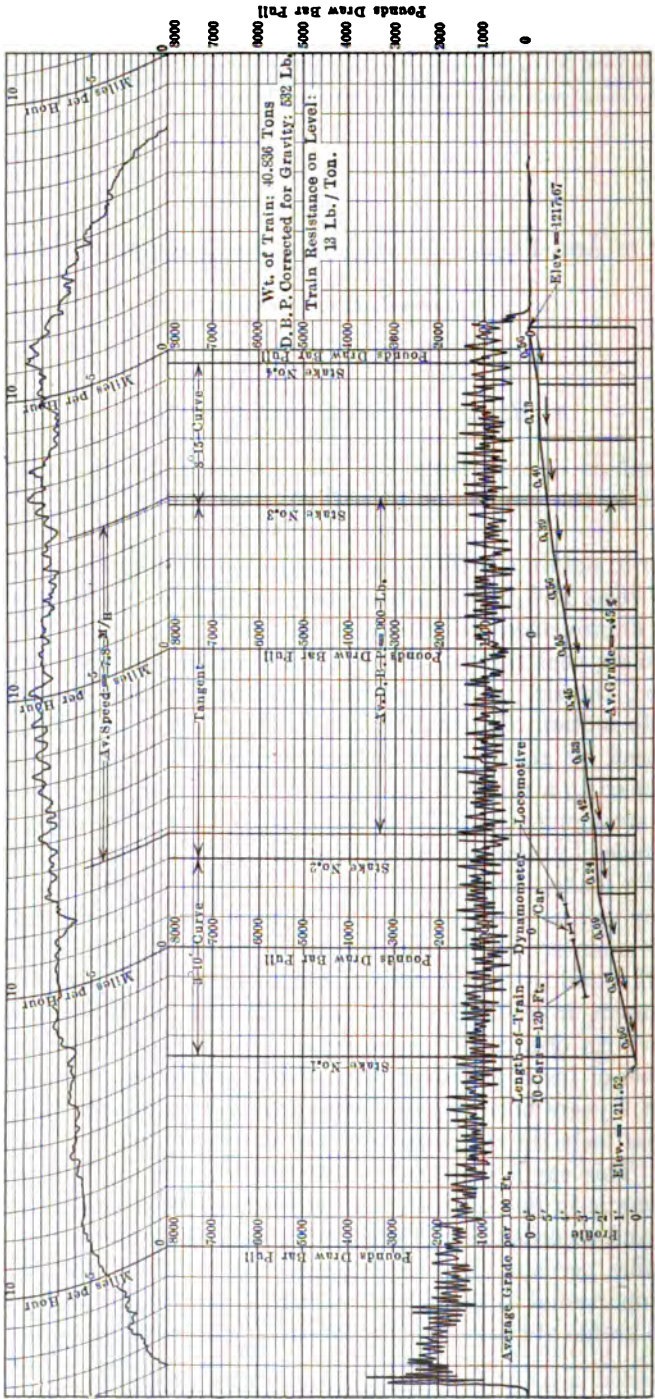


FIG. 1.—TEST ON 10 LOADED FLEXIBLE ROLLER-BEARING MINE CARS.

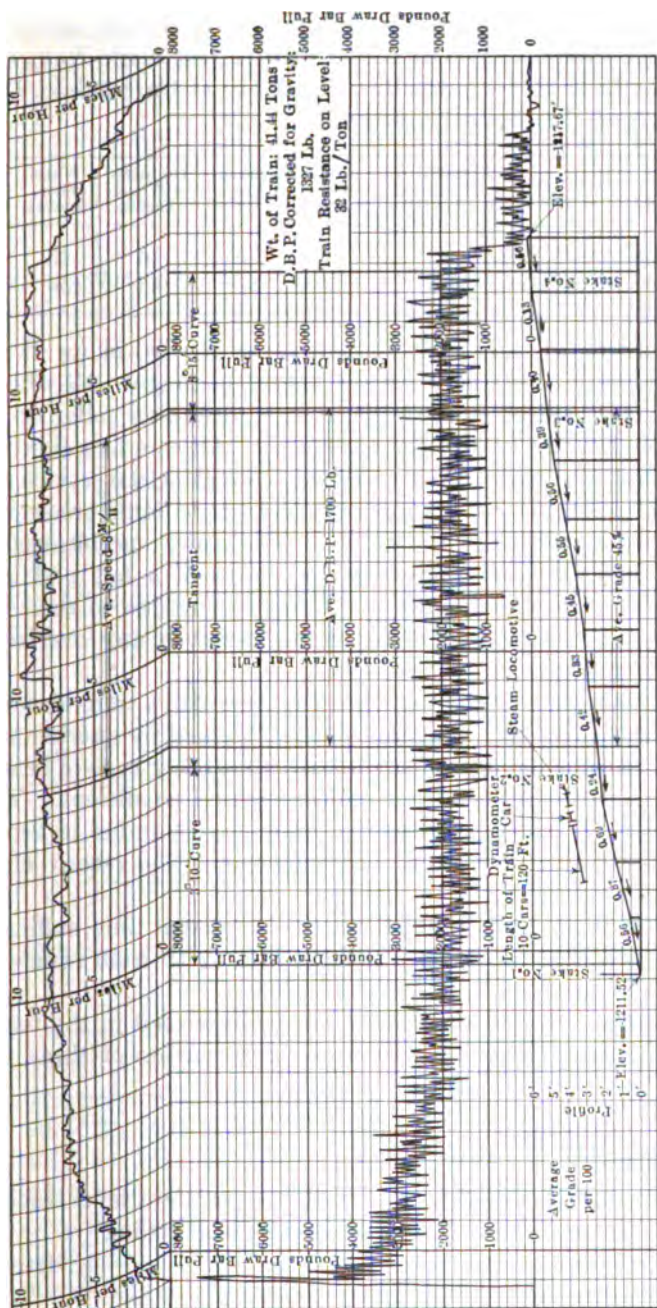


FIG. 2.—TEST ON 10 LOADED PLAIN-BEARING MINE CARS.



regarding train resistance for any grade or piece of track can be read from the chart at a glance. As an illustration, two charts are shown in Figs. 1 and 2 that were obtained at a comparative test made Nov. 17 to 20, 1916, at the D. & H. Co.'s Coalbrook Colliery at Carbondale, Pa. The following types of mine cars were used:

| Item No. | Type of Bearing | Location of Bearing | Diameter of Journal, Inches | Length of Bearing, Inches | Track Gage, Inches | Diameter of Wheel, Inches | Wheel Base, Inches | Drawbar to Top of Rail, Inches |
|----------|-----------------|---------------------|-----------------------------|---------------------------|--------------------|---------------------------|--------------------|--------------------------------|
| 1        | Roller          | Outside box         | 2 $\frac{7}{8}$             | 4                         | 30                 | 16                        | 36                 | 14                             |
| 2        | Plain           | Outside box         | 2 $\frac{7}{8}$             | 4                         | 30                 | 16                        | 36                 | 14                             |

These were wooden cars built by the D. & H. Co. in its own shops. They were identical in every respect except bearings. All wheels were loose and were held on the axle between a shrunk-on collar on the one side and the bearing box on the other side. All cars had side bumpers. When the test was made the cars had seen enough actual service to insure well run-in bearings. For lubrication a high-grade liquid grease was used compounded according to the D. & H. Co.'s own specifications. The plain bearings were self-lubricating by means of a felt pad in an oil reservoir; the roller bearings were self-lubricating due to their inherent hollow helical construction. No bearings received any overhauling or special treatment for any of the tests.

Each one of the various tests was run first with a train of cars having one type of bearing and was immediately followed by another test with a train of the same number of cars but with the other type of bearing and on the same piece of track and under identical conditions. This particular series of tests was witnessed by a large number of representative anthracite mine operators and engineers. The record charts are self-explanatory.

From the many tests conducted up to the present time, sufficient data have been collected to obtain a definite idea as to train resistance and its relation to speed.

An average value of these data can be expressed in the form of an equation. Any formula for train resistance consists of three terms as follows:

$$R = A + BS + CS^2$$

in which

$R$  = train resistance in pounds per ton on straight level track.

$A$  = friction constant relating to bearing and rolling friction.

$B$  = friction constant relating to speed.

$C$  = friction constant relating to air resistance.

$S$  = speed in miles per hour.

} determined  
by  
experiment.



On account of the comparatively low speeds encountered in mine service, the air friction is insignificant, and therefore the last term of the above formula,  $CS^2$ , can be dropped. The test data were found to meet the following equations:

$R_p = 26 + 0.45S$  = train resistance of loaded plain-bearing cars in pounds per ton.

$R_r = 9 + 0.45S$  = train resistance of loaded flexible roller-bearing cars in pounds per ton.

These formulas apply to average mine conditions, with clean tracks, with cars of from 2 to 4-ton capacity, and with wheels not below 16 in. in diameter.

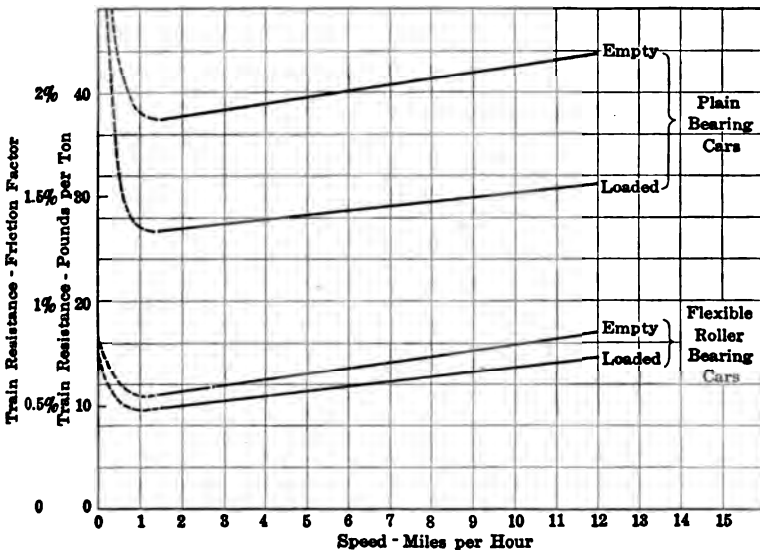


FIG. 3.—RELATION BETWEEN TRAIN RESISTANCE AND SPEED OF COAL-MINE CARS (AVERAGE FIGURES).

Tracks covered with coal dust were found to cause an increase in the train resistance of about 30 per cent. for either type of bearing. It should be remembered that this is the figure obtained with the dynamometer car behind the locomotive and that the locomotive, by rolling down the dust and loose coal particles, lessens to a large extent their effect on the train of cars behind it. The electrical input into the locomotive will, of course, be considerably increased over the 30 per cent., but this additional loss was not measured during these tests.

Tests made with empty cars showed that the train resistance per ton is about 40 per cent. higher for plain-bearing cars and about 15 per cent. higher for the roller-bearing cars than would be obtained from the formula for loaded cars.

Fig. 3 shows graphically the relation between train resistance - speed of loaded and empty cars. The train resistance is given both in pounds per ton and as friction factor. The friction factor is simply the percentage of the train weight required to keep the train in motion at any given speed.

If  $W_L$  = the weight of a train of loaded cars in tons and  $W_E$  = the weight of a train of empty cars in tons, the total train resistance in pounds on straight level track will be:

$$F_L = W_L \times R_L \text{ and } F_E = W_E \times R_E$$

When running on a grade the effect of gravity is taken into consideration as follows:

$$F_L = (W_L \times R_L) \pm (W_L \times G \times 20) \text{ and } F_E = (W_E \times R_E) \pm (W_E \times G \times 20)$$

$G$  is the grade in per cent., while 20 is the added drawbar pull in pounds per ton per 1 per cent. of grade.

The grade also affects the tractive effort of the locomotive by the grade percentage of its own weight. If  $M$  = the weight of the locomotive in tons, then, for a loaded train of cars,

$$F_L \pm (M \times G \times 20) = (W_L \times R_L) \mp (W_L \times G \times 20)$$

and for a train of empty cars,

$$F_E \mp (M \times G \times 20) = (W_E \times R_E) \pm (W_E \times G \times 20)$$

The symbols have the following meaning:

$F_L$  = total train resistance of loaded cars in pounds.

$F_E$  = total train resistance of empty cars in pounds.

$W_L$  = weight of a train of loaded cars in tons.

$W_E$  = weight of a train of empty cars in tons.

$R_L$  = train resistance of loaded cars in pounds per ton.

$R_E$  = train resistance of empty cars in pounds per ton.

$M$  = weight of locomotive in tons.

$G$  = grade in per cent.

20 = gravity force in pounds per ton per 1 per cent. of grade.

## 2. Test Data Commercially Applied

The following comparisons are based on the ideal case of two mines, one equipped throughout with plain-bearing cars and the other with roller-bearing cars, each mine being laid out to the theoretical grade that will make the locomotive effort required to pull a train of loaded cars down grade equal to the effort required to pull the same train of empties against the grade. Such an example can easily be altered to

apply to any individual mine by simply making the necessary change of those items which differ from the example.

A 1,000-ton-per-day mine will be considered based on the viewpoint of reducing the cost of mining 1,000 tons of coal a day.

Some mine operators are also interested to know the quickest method of development or increase of output of their mines by equipping their cars with flexible roller bearings with no increased overhead except the additional wages required for the extra miners. An increased output is naturally limited by the length of landings, the accommodation of extra miners, the capacity of the gathering and haulageways, the capacity of cage and tipple, etc. There are too many different possibilities to make the working out of an example possible, but, fortunately, every mining operator will be able to determine for his individual case the possible increase of output by utilizing the figures and test data described above.

The following is a description of the 1,000-ton-per-day mine:

|   |                   |
|---|-------------------|
| Output of coal per day.....                   | 1,000 tons        |
| Capacity of each mine car.....                | 3 tons            |
| Number of car loads to supply 1,000 tons..... | 333 $\frac{1}{3}$ |
| Weight of an empty car.....                   | 2,500 lb.         |
| Weight of a loaded car.....                   | 8,500 lb.         |
| Average distance per car per trip loaded..... | 5 miles           |
| Average distance per car per trip empty.....  | 5 miles           |
| Average speed of trains.....                  | 6 miles per hour  |
| Working hours per day.....                    | 8                 |
| Time for a round trip, hours.....             | 2                 |
| Round trips per day per train.....            | 4                 |

To find the theoretical grade that will result in equalized locomotive effort in pulling a loaded train down grade and bringing the same train of empty cars up the grade, that is,  $F_L = F_E = F$ , the last two equations can be written as follows:

$$\frac{F - (M \times G \times 20)}{(W_E \times R_E) + (W_E \times G \times 20)} = \frac{F + (M \times G \times 20)}{(W_L \times R_L) - (W_L \times G \times 20)}$$

it then follows:

$$G = \frac{(W_L \times R_L) - (W_E \times R_E)}{20(W_E + W_L + 2M)}$$

In these equations the designations given below are used:

- $F$  = drawbar pull of locomotive on level in pounds.
- $M$  = weight of locomotive in tons.
- $G$  = percentage of the desired grade.
- $W_E$  = weight of a train of empty cars in tons.
- $W_L$  = weight of a train of loaded cars in tons.
- $R_E$  = train resistance of empty cars in pounds per ton.
- $R_L$  = train resistance of loaded cars in pounds per ton.
- 20 = gravity force in pounds per ton per 1 per cent. of grade.

Substituting for the train resistance the figures found by test for speed of 6 miles per hour, the data shown in Table 1 are obtained:

TABLE 1

|   | Plain-bearing<br>Cars | Roller-bearing<br>Cars |
|---|-----------------------|------------------------|
| Train resistance of loaded cars, pounds per ton.....                            | 28.6                  | 11.6                   |
| Train resistance of empty cars, pounds per ton.....                             | 40.0                  | 13.6                   |
| Grade for equalized drawbar pull, per cent.....                                 | 0.61                  | 0.284                  |
| Number of cars per trip.....  | 42                    | 42                     |
| Weight of coal per train, tons.....   | 126                   | 126                    |
| Train resistance, loaded cars out, pounds.....                                  | 2,930                 | 1,069                  |
| Train resistance, empty cars in, pounds.....                                    | 2,740                 | 1,027                  |
| Train resistance, starting (estimate), pounds.....                              | 4,000                 | 1,200                  |
| Necessary weight of locomotive, tons.....                                       | 8                     | 3                      |
| Number of locomotives.....  | 2                     | 2                      |
| Foot-pounds per train per round trip.....                                       | 146,800,000           | 54,100,000             |
| Kilowatt-hours per train per round trip.....                                    | 55.3                  | 20.4                   |
| Kilowatt-hours all trips per day.....   | 443                   | 163                    |
| Foot-pounds per locomotive per round trip.....                                  | 8,430,000             | 3,160,000              |
| Kilowatt-hours per locomotive per round trip.....                               | 3.180                 | 1.195                  |
| Kilowatt-hours of two locomotives per day.....                                  | 25.40                 | 9.55                   |
| Total kilowatt- hours per day of cars and locomotive.....                       | 468.40                | 172.55                 |
| Total cost of power per day at 2 c. per kilowatt-hour at locomotive*.....       | \$9.36                | \$3.45                 |
| Total cost of power per year of 300 days.....                                   | \$2,810.00            | \$1,035.00             |
| Lubrication material and labor per car per year as found in actual service..... | \$1.73                | \$0.29                 |
| Lubrication 164 cars per year.....  | \$284.00              | \$47.50                |
| Total cost of power and lubrication per year.....                               | \$3,094.00            | \$1,082.50             |
| Saving.....   |                       | \$2,011.50             |

\* The power cost of 2 c. per kilowatt-hour at the locomotive corresponds to a charge of approximately 1 c. per kilowatt-hour at the switchboard. This is due to the losses taking place in transformers, converters, transmission line, locomotive motor, rails, rail bonding, etc.

A clear saving of \$2,011.50 per year has been accomplished through the application of flexible roller-bearings. These roller-bearing cars are necessarily more expensive than the ordinary plain-bearing cars. The 164 cars in the above example would require an additional investment of about \$20 per car, or \$3,280 based on prices prevailing at the present time. This amount is balanced by the saving in the size of the locomotives. Each one of the 8-ton locomotives costs about \$1,000 more than one of the 3-ton locomotives, resulting in a saving of about \$2,000. The 8-ton locomotive would require for the track rails weighing 30 lb. per yard, while the 3-ton locomotive could get along with 20-lb. rails. The difference in cost amounts to about \$3,500 for the example based on \$40 per ton and 5 miles of track.

The last item alone has wiped out the extra cost of the roller-bearing cars. On top of this should be figured the saving accomplished in the electrical transmission line, transformers, power house, reduced derailments, reduced wear and tear on cars and tracks, reduced size of repair-shop, etc.

Inasmuch as it is possible only in new mines and in a limited number of existing mines to arrange the haulage road for an ideal grade, the above example shall be supplemented with another example of a mine having a prevailing grade of, say, 3 per cent. in favor of load, and still another example will be given based on a prevailing grade of 3 per cent. against load.

Based on the same output, car weights, number of trips, etc., as before, the figures for the 3 per cent. grade in favor of load turn out as shown in Table 2.

TABLE 2

|  | Plain-bearing<br>Cars | Roller-bearing<br>Cars |
|--|-----------------------|------------------------|
| Train resistance of loaded cars, pounds per ton.....                     | 28.6                  | 11.6                   |
| Train resistance of empty cars, pounds per ton.....                      | 40.0                  | 13.6                   |
| Grade against empty, per cent.....                                       | 3                     | 3                      |
| Number of cars per trip.....   | 28                    | 28                     |
| Weight of coal per train, tons.....                                      | 84                    | 84                     |
| Train resistance going in, pounds.....                                   | 3,500                 | 2,575                  |
| Train resistance starting, going in, pounds.....                         | 4,200                 | 2,660                  |
| Train resistance going out, pounds.....                                  | -3,740                | -5,760                 |
| Necessary weight of locomotive, tons.....                                | 13                    | 8                      |
| Number of locomotives.....   | 3                     | 3                      |
| Foot-pounds per train per round trip.....                                | 92,500,000            | 68,000,000             |
| Kilowatt-hours per train per round trip.....                             | 34.9                  | 25.6                   |
| Kilowatt-hours all trips per day.....                                    | 419                   | 307                    |
| Foot-pounds per locomotive per round trip.....                           | 27,500,000            | 16,900,000             |
| Kilowatt-hours per locomotive per round trip.....                        | 10.35                 | 6.37                   |
| Kilowatt-hours of three locomotives per day.....                         | 124.0                 | 76.5                   |
| Total kilowatt-hours per day of locomotive and train...                  | 543.0                 | 383.5                  |
| Total cost of power per day at 2 c. per kilowatt-hour at locomotive..... | \$10.86               | \$7.67                 |
| Total cost of power per year of 300 days.....                            | \$3,260               | \$2,300                |
| Lubrication, material and labor per car per year.....                    | \$1.73                | \$0.29                 |
| Lubrication 164 cars per year.....                                       | \$284.00              | \$47.50                |
| Total cost of power and lubrication per year.....                        | \$3,544.00            | \$2,347.50             |
| Saving.....  |                       | \$1,196.50             |

To the saving of \$1,196.50 should be added the saving due to the reduced size of locomotives and to the reduction in the weight of the rails. Each one of the 13-ton locomotives costs about \$1,200 more than one of the 8-ton locomotives, resulting in a saving of \$3,600. The 13-ton

locomotive would require rails weighing 50 lb. per yard against the 8-ton locomotives using 30-lb. rails which accounts for a saving of \$7,000.

The figures for the 3 per cent. grade against load are shown in Table 3.

TABLE 3

|  | Plain-bearing<br>Cars | Roller-bearing<br>Cars |
|--|-----------------------|------------------------|
| Train resistance of loaded cars, pounds per ton.....                     | 28.6                  | 11.6                   |
| Train resistance of empty cars, pounds per ton.....                      | 40.0                  | 13.6                   |
| Grade against load, per cent.....  | 3                     | 3                      |
| Number of cars per trip.....   | 14                    | 14                     |
| Weight of coal per train.....  | 42                    | 42                     |
| Train resistance going out, pounds.....                                  | 5,270                 | 4,260                  |
| Train resistance starting, going out, pounds.....                        | 7,460                 | 4,520                  |
| Train resistance going in, pounds.....                                   | -350                  | -812                   |
| Necessary weight of locomotives, tons.....                               | 25                    | 15                     |
| Number of locomotives.....   | 6                     | 6                      |
| Foot-pounds per train per round trip.....                                | 278,000,000           | 225,000,000            |
| Kilowatt-hours per train per round trip.....                             | 104.5                 | 85.0                   |
| Kilowatt-hours all trains per day.....                                   | 1,255                 | 1,020                  |
| Foot-pounds per locomotive per round trip.....                           | 52,800,000            | 31,600,000             |
| Kilowatt-hours per locomotive per round trip.....                        | 19.9                  | 11.9                   |
| Kilowatt-hours of six locomotives per day.....                           | 119.5                 | 71.5                   |
| Total kilowatt-hours per day of locomotives and trains...                | 1,375                 | 1,092                  |
| Total cost of power per day at 2 c. per kilowatt-hour at locomotive..... | \$27.50               | \$21.80                |
| Total cost of power per year of 300 days.....                            | \$8,250               | \$6,540                |
| Lubrication, material and labor per car per year.....                    | \$1.73                | \$0.29                 |
| Lubrication 164 cars per year.....                                       | \$284.00              | \$47.50                |
| Total cost of power and lubrication per year.....                        | \$8,534.00            | \$6,587.50             |
| Saving.....  |                       | \$1,946.50             |

Due to the reduction in the size of locomotives from 25 tons to 15 tons, a saving of at least \$1,500 per locomotive, or \$9,000 for the six locomotives, has been accomplished.

The lighter locomotives need 50-lb. rails against the 70-lb. rails of the heavier locomotives. This difference in weight accounts for a saving of about \$7,000.

In determining the size of locomotives, no attention has been paid to the size of the tunnels. If any such locomotive should be of too large a size, then two smaller locomotives would have to be substituted, each locomotive pulling half the number of cars that could have been hauled by the bigger locomotive. The two smaller locomotives would cost more than one big locomotive. Besides this, there is the expense of the motorman and helper for the extra train. It is obvious that this handicap is particularly great with plain-bearing cars.

The main items are tabulated again for easy comparison:

## SUMMARY

*Ideal Grade:*

|   |            |
|---|------------|
| Saving in investment for locomotives.....   | \$2,000.00 |
| Saving in investment for rails.....         | 3,500.00   |
| <hr/>                                       |            |
| Total saving in investment.....             | 5,500.00   |
| Investment for roller bearings.....         | 3,280.00   |
| <hr/>                                       |            |
| Net saving in investment.....               | 2,220.00   |
| Yearly saving in power and lubrication..... | \$2,011.50 |

*3 Per Cent. Against Empties:*

|   |            |
|---|------------|
| Saving in investment for locomotives.....   | \$3,600.00 |
| Saving in investment for rails.....         | 7,000.00   |
| <hr/>                                       |            |
| Total saving in investment.....             | 10,600.00  |
| Investment for roller bearings.....         | 3,280.00   |
| <hr/>                                       |            |
| Net saving in investment.....               | 7,320.00   |
| Yearly saving in power and lubrication..... | \$1,196.00 |

*3 Per Cent. Against Load:*

|   |            |
|---|------------|
| Saving in investment for locomotives.....   | \$9,000.00 |
| Saving in investment for rails.....         | 7,000.00   |
| <hr/>                                       |            |
| Total saving in investment.....             | 16,000.00  |
| Investment for roller bearings.....         | 3,280.00   |
| <hr/>                                       |            |
| Net saving in investment.....               | 12,720.00  |
| Yearly saving in power and lubrication..... | \$1,946.50 |

For obvious reasons, these examples had to be reduced to the simplest possible conditions, and therefore the hauling in of timbers and supplies and the hauling out of rock were not taken into consideration in either case and no allowance has been made for the increased train resistance on curves, it being assumed that only curves of liberal radius were employed and that the grade was sufficiently eased up at the curves to compensate for the increased drawbar pull.

These items and others, the acceleration of the trains, for instance, will of course increase the total power consumption in each case, but they will not materially affect the relative proportion of the three examples.

The examples bring out forcibly the importance of proper grade. It has been clearly established that while considerable economy can be obtained through the introduction of roller bearings on existing grades, yet the maximum saving in the total expenditures for haulage can be obtained only by a proper arrangement of grades as shown in the first example. When heavy grades occur they are in most cases only of short length and with roller-bearing cars they can be taken flying, that is, the momentum of the moving train will carry the train over such grades.

It is safe to say that the flexible roller bearing will do more for the

mine car than it has done for the modern automobile, because, besides its durability and certainty of operation, there are other features which make it particularly attractive and profitable. To begin with, there is the saving in power which accounts for a smaller power bill, smaller locomotives, smaller transmission line and power house, lighter rails, less expensive rail bondings, less wear and tear on cars and tracks with fewer derailments, greater safety of operation, and steady and uninterrupted production. It will be noticed that only such items as directly affect the haulage cost and can be expressed in dollars and cents have been figured. There are other features, however, which cannot readily be figured, such as the effect on mules of easily running cars, where cars are gathered by mules, or the effect on the miners where cars have to be moved by hand. This is important in itself, as the human element more and more demands the elimination of all useless drudgery. The effect is very pronounced, too, when storage-battery locomotives are used.

The merits of storage-battery locomotives for gathering purposes and for short hauls are generally appreciated. Their battery capacity is necessarily limited and it is highly desirable that one charge of the battery be sufficient for one day's run. The two things affecting the battery capacity are grade and car journal friction. The grade quickly limits the use of storage-battery locomotives, while bearing friction places a serious handicap on their productiveness. A glance at the curves showing relative train resistance and a comparison of the kilowatt-hours given in the above examples will prove that a storage-battery locomotive with a single charge per day will be able to haul considerably more coal when flexible roller-bearing cars are used in place of plain-bearing cars. By the use of these figures it should be easy to figure out the possible production of any individual mine when full particulars regarding grade, length of haul, etc., are known. Not to be overlooked should be the heavy current discharges when starting a train of plain-bearing cars. These high starting peaks seriously affect the capacity and life of a storage battery. A locomotive pulling roller-bearing cars will not be subjected to such peak loads; on the contrary, it will pick up faster and consequently gather faster.



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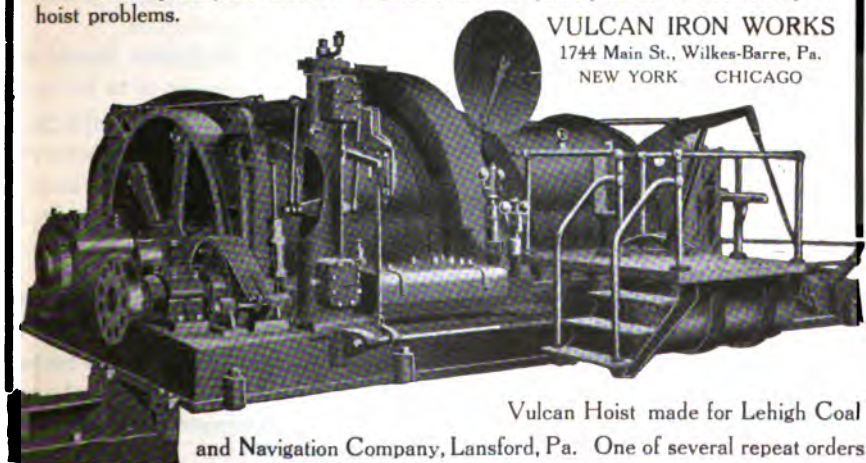
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### SODIUM HYDROXIDE IN BOILER FEED WATER CAUSES TROUBLE

In certain districts where feed water for boilers contains sodium hydroxide, many boiler troubles have appeared. Such water is found in the central eastern part of Illinois, in the Fox River valley in the northern part of the same State, and in portions of other States.

The Engineering Experiment Station of the University of Illinois has just completed an investigation of this source of boiler distress, and the results are published in Bulletin 94 by S. W. Parr.

It was noted that boilers using feed water containing sodium hydroxide often developed fine cracks radiating from rivet holes or extending from hole to hole. The experiments showed that the effect upon the metal is to cause brittleness which makes it less capable of withstanding steam pressure and temperature changes. Among the remedies suggested is a salt which has properties that cause it to react with the alkali and yield a harmless product.

Copies of Bulletin No. 94 may be obtained free of charge from C. R. Richards, Director, Urbana, Illinois.

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### Trade Catalogs

The Merchant & Evans Co. has issued a small folder dealing with "Almetl" fire doors and shutters. These are said to be impassable by fire, approved and labeled by Underwriters, indestructible, easy to operate. They are made of steel, with welded joints.

The Vulcanite Portland Cement Co., in a small pamphlet issued on May 1, advocates the use of bulk cement where the cement is to be used from car to mixer. Bulk cement can be measured into a wheelbarrow by the use of a bottomless box, as sand or crushed stone is measured. It packs evenly in the car, nails in the car floor cannot injure it, and it compares very favorably with sack cement in respect to dust. It is economical because sack losses and expenses are saved.

The Brinell Meter, a portable apparatus for the accurate determination of the Brinell hardness of metals and metal products, is described in a folder issued by Herman A. Holz, 50 Church St., New York. This meter was invented by F. H. Schoenfuss, Director of the Metallurgical Laboratory, Standard Roller Bearing Co. It is said to eliminate the sampling error, because no cutting of specimens is required.



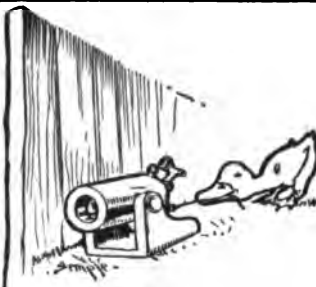
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The General Electric Co. has issued Bulletins 48900 and 44424. Bulletin 48900 is on CR-9510 Automatic Solenoid Brakes and CR-9500 Solenoids. The bulletin contains ordering directions, and complete dimensions of the various kinds of solenoids and brakes. No. 44424 deals with GE-203-P Ventilated Commutating Pole Motors.

The Tungar Rectifier is a new battery charger for use with alternating current, that has been developed by the General Electric Co.'s Research Laboratory. Some of its advantages as listed by the manufacturers are: low cost, high efficiency, self starting, no moving parts, no fire risk, no floor space.


Another leaflet from the same company describes small battery-charging outfits for charging ignition and lighting batteries.

The Denver Fire Clay Co. has a folder on the Case Low Pressure Oil Forge for heating drill steel. In this forge the drills are under observation at all times. It is said to be almost noiseless, on account of the low pressure used, and to be so regulated that complete combustion takes place, which eliminates smoke and the forming of carbon in the forge.

Catalogue 3002, Westinghouse Electric & Manufacturing Co., covering industrial apparatus, is being issued in a new form. It will be divided into five separately bound sections, 8½ by 11 in., which will be known by the following titles:

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| Westinghouse Direct-Current Motors and Generators | Cat. 30 |
| Westinghouse Alternating-Current Motors.....      | Cat. 33 |
| Westinghouse Control Apparatus.....               | Cat. 36 |
| Westinghouse Small Motors and Generators.....     | Cat. 38 |
| Westinghouse Mill, Crane and Hoist Motors.....    | Cat. 45 |

Catalogue 38 is the first of these catalogs to be distributed. It contains complete descriptions, ratings and dimensions of small motor apparatus, including small power motors, types CA, CAH, AR, CSA, and CW, Ventura Fans, Sirocco Blowers, Sewing Machine Motors, Generators, Buffing Motors, Small Motor-Generators and Small Motor Accessories.



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PUBLISHED MONTHLY

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Signed.....

Address.....

.....

# AMERICAN INSTITUTE OF MINING ENGINEERS

29 West 39th Street, New York, N. Y.

## PROPOSAL FOR MEMBERSHIP

Mr. \_\_\_\_\_

Address \_\_\_\_\_

is hereby proposed by the undersigned, as a

|   |               |
|---|---------------|
| { | Member        |
|   | Associate     |
|   | Junior Member |

of the American Institute of Mining Engineers.

|                                  |   |  |
|----------------------------------|---|--|
| _____<br>_____<br>_____<br>_____ | { | Signatures of<br>three Members<br>or Associates. |
|----------------------------------|---|--|

Place of birth \_\_\_\_\_ Year of birth \_\_\_\_\_

Education, when, where and how acquired, with degrees, if any.  
*A technical education is not requisite for membership.*

| Dates |  |
|-------|--|
|       |  |
|       |  |
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|       |  |

Please give record of experience on following page with dates; for example,  
in the form used by "Who's Who In America" and similar publications.

[illegible]

(If the applicant has previously been proposed as a Member, Associate or Junior Member of the Institute this fact should be stated.)

**Signature of Applicant:** \_\_\_\_\_

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

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**BUCYRUS REVOLVING STEAM SHOVEL, NEAR MULBERRY, KANS.,  
AUGUST, 1916, 80-FT. BOOM, 7-YD. DIPPER, WEIGHT, 300 TONS.**



**SULLIVAN MACHINE, INDIAN CREEK COAL CO., NEAR BICKNELL, IND.**

*(Frontispiece.)*

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# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 128

AUGUST

1917

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Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

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## WAR NOTICE

Attention!

Engineers!

Geologists!

*Here is your Opportunity to be of Professional Service*

The war has interrupted the supply of many minerals which were normally imported. In view of present transportation conditions, a serious shortage exists in certain materials and will soon develop in others.

Some immediate action is imperative. The first thing is to take a careful inventory of American resources. We must know where the ore deposits are, what has hitherto hindered their utilization, what methods must be used, and what equipment will be needed if an emergency demands their rapid development.

Many properties must be re-examined from a new point of view, that of present necessity rather than of normal profit. When the shortage is realized, the preliminary steps for relief should have been completed. It is a necessary and patriotic form of mineral insurance. Will you have a part in it?

Do not forget that these are war times, and that what may have been valueless a year ago may be a "life-saver" now; an abandoned copper prospect of 1913 might make a successful pyrite mine in 1917. We are fighting an autocratic enemy who knows his resources to the last pound, and uses them with most efficient organization. We have no Kaiser, so each of us must do his share to put our Government in the same situation. The Government needs your facts.

Part of this essential information is contained in the mining reports in

your files. The following minerals are of special importance: Pyrite and Pyrrhotite east of the Rocky Mountains, Sulphur, Manganese and Manganiferous Iron Ore, Lead, Nitrate, Chromite, Tin, Nickel, Platinum, Flake Graphite, Sheet Mica, and high grade Refractory Clay.

Will you submit a list of the reports (or the reports themselves, if you prefer) which you have on properties and deposits which may be of importance under the present abnormal war conditions? Give the date of the report, the name of the engineer who made it, the type of ore, the precise location of the deposit, and its relation to transportation. On request, will you loan these reports for the use of the War Minerals Committee, or such appropriate Department of the Government as you may specify? Reports will be held confidential if you so indicate. Can you donate a certain amount of service to the further examination of properties which may be selected as of greatest importance?

Address: The War Minerals Committee,  
Room 4241, Interior Department Building,  
Washington, D. C.

The members of this committee who have been designated by their respective organizations are as follows:

William Young Westervelt, Consulting Mining Engineer, Chairman, representing the American Institute of Mining Engineers, and the Mining and Metallurgical Society of America.

Alfred G. White, Mine Economist, United States Bureau of Mines.

W. O. Hotchkiss, State Geologist of Wisconsin, representing the Geology Committee of the National Research Council and the Association of State Geologists.

David White, Chief Geologist, United States Geological Survey.

This committee has, as its specific purpose, the consideration of the development of a sufficient supply of those minerals of which a scarcity now exists or may develop. It is intended to promote united action by the mining and geological professions as represented by Federal and State bureaus and the mining societies; and to secure the effective service of such individual engineers and geologists as desire to be of assistance and who patriotically volunteer their services.

NOTE.—If members prefer to forward reports or information through the headquarters of the Institute, the Secretary's office will gladly serve as intermediary.

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## ST. LOUIS MEETING

The visitors to the October meeting will be given opportunity to visit the coal fields in Illinois, adjacent to St. Louis, the steam-shovel operations near Pittsburgh, Kan., and the Oklahoma oil fields.

The coal mines of Illinois, Missouri, Kansas, Oklahoma and Arkansas furnish fuel over all of the central region of the country. Especially at this time, this district is of importance because of the necessity of producing enormous quantities to meet the country's demand. Some of the mines produce as much as 5,500 tons in 8 hr., and the total output of Illinois, alone, in 1916, was 63,000,000 tons. Visitors may see elaborate underground electric tram systems, large underground machine repair shops, surface plants equipped for rapid loading, and some of the most modern coal-washing installations.

The conditions for mining here are somewhat simpler than in the east.



There are practically no steep grades; the coal lies nearly flat and within 300 to 600 ft. of the surface.

The Pittsburg, Kan., field is of peculiar interest, in that there are over 50 steam shovels, many of 8 cu. yd. capacity, stripping the overburden to lay bare the coal.

The oil and gas resources are worthy of note. From the gas wells in western Illinois, some of which yield 20,000,000 cu. ft. a day, natural gas is now piped as far as East St. Louis.

The premier oil fields of America are in Oklahoma, but 425 miles from St. Louis. The production of these fields, in 1916, was 106,396,834 barrels, valued at \$248,735,172. This represents 35 per cent. of the production of the United States, and means 290,700 barrels of oil an hour. Kansas produced 13,961,000 barrels. The gasoline production in Oklahoma exceeds 1,000,000,000 gallons per year. There are more than 1400 casing head gasoline plants in operation in Oklahoma, and 74 refineries in Kansas and Oklahoma.

In these two states some 81,000 wells have been drilled, of which 43,000 were oil wells and something less than one-third of the remainder were gas wells. Of the 43,000, 6000 have an average production of about 48 barrels per day, the others from  $\frac{1}{2}$  to 2 barrels a day.

Among the best known pools are: the Glenn, covering 20,000 acres; the Cushing, covering 18,000 acres, which produced as much as 325,000 barrels a day; the Healdton, covering 8,200 acres, having a daily production of 55,000 barrels; the Kennedy-Springer, covering 5000 acres, with a daily production of 30,000 barrels.

The Cosden Refinery at West Tulsa is one of the largest in the country, with a daily capacity of 40,000 barrels crude. The 71 independent refineries have a daily capacity of 150,000 barrels.

More than 500 oil geologists and petroleum engineers are engaged in exploration and intensive investigations, and new pools are being developed.

The gas resources of this district are material, the annual production being somewhat in excess of 100,000,000 cu. ft.

J. D. ROBERTSON,  
*Chairman, Publicity Committee.*

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## INVITATION TO ROLLA

The Curators and Faculty of the Missouri School of Mines cordially invite the members of the American Institute of Mining Engineers to visit the School of Mines at Rolla during the St. Louis meeting in October, 1917. On arrival, members are requested to register with the Director, Dr. A. L. McRae.

## COMMITTEE CHAIRMEN FOR THE ST. LOUIS MEETING

GENERAL COMMITTEE—H. A. Buehler, Rolla, Missouri

BOOK COMMITTEE—F. W. DeWolf, Urbana, Illinois

ENTERTAINMENT COMMITTEE—James E. Caselton, Saint Louis

EXCURSIONS COMMITTEE—A. H. Wheeler, Saint Louis

FINANCE COMMITTEE—E. F. Goltra, Saint Louis

## JOPLIN-WEBB CITY-MIAMI COMMITTEES

*General*, Victor Rakowsky*Joplin*, F. C. Wallower*Webb City*, W. V. Shackelford*Miami*, J. F. Robinson

[LADIES' ENTERTAINMENT—Mrs. P. N. Moore, Saint Louis

PITTSBURG, KANSAS, COMMITTEE—J. F. Klaner, Pittsburgh, Kansas

H. N. Taylor, Kansas City, Missouri

PUBLICITY COMMITTEE—James D. Robertson, Carlinville, Illinois

[RECEPTION COMMITTEE—P. N. Moore, Saint Louis

SOUTHEAST MISSOURI COMMITTEE—C. J. Adami, Bonne Terre, Missouri

TRANSPORTATION COMMITTEE—Eugene McAuliffe, Paducah, Ky.

TULSA COMMITTEE—L. L. Hutchison

*Publicity*, J. B. Newby*Finance*, N. K. Moody*Arrangements and Transportation*, M. J. Munn*Entertainment and Reception*, D. Hager; Col. C. B. Douglas, *Honorary Chairman*

## RATES OF HOTELS IN ST. LOUIS

## PLANTERS HOTEL

Location, Fourth and Pine Streets. This hotel will be the headquarters for the meeting. Rooms available, between 300 and 400.

|                           | Rate Per Day |                         |
|---------------------------|--------------|-------------------------|
| Single, with bath.....    | \$2.00,      | \$2.50 and \$3.00       |
| Double, with bath.....    | 3.00,        | 3.50, \$4.00 and \$5.00 |
| Single, without bath..... | 1.50,        | 2.00                    |
| Double, without bath..... | 2.00,        | 2.50, 3.00 and 4.00     |

## AMERICAN HOTEL AND ANNEX

Location, Seventh and Market Streets. Rooms available, American, 200; Annex 150.

|             |        |
|-------------|--------|
| Single..... | \$1.50 |
| Double..... | 1.25   |

All rooms have either private tub or shower bath and circulating ice water.

## BUCKINGHAM HOTEL

Location, Kingshighway and West Pine Boulevards. Rooms available for 40 persons.

|   |        |
|---|--------|
| No single rooms with bath.                              |        |
| Double suite, with bath between 4 persons to suite..... | \$1.50 |
| Single without bath (1 person).....                     | 2.00   |
| Single without bath (2 persons).....                    | 3.00   |

## JEFFERSON HOTEL

Location, Twelfth and Locust Streets. Rooms available, 100 or more.

|                           |               |
|---------------------------|---------------|
| Single, with bath.....    | \$2.50 and up |
| Double, with bath.....    | 4.00 and up   |
| Single, without bath..... | 2.00          |
| Double, without bath..... | 3.00          |

**MAJESTIC HOTEL**

Location, Eleventh and Pine Streets. Rooms available, 100, accommodating, approximately 150 persons.

|  |                           |
|--|---------------------------|
| Single.....                                      | \$1.50, \$2.00 and \$2.50 |
| Double.....                                      | 2.50, 3.00 and 3.50       |
| Private bath in each room, either tub or shower. |                           |

**MARQUETTE HOTEL**

Location, Eighteenth Street and Washington Avenue. Rooms available, 200 to 300.

|                           |                           |
|---------------------------|---------------------------|
| Single, with bath.....    | \$1.50, \$2.00 and \$2.50 |
| Double, with bath.....    | 3.00 and up               |
| Single, without bath..... | 1.00 and \$1.50           |
| Double, without bath..... | 2.00                      |

Rooms without bath have running water.

**MARYLAND HOTEL**

Location, Ninth and Pine Streets.

|                           |                         |
|---------------------------|-------------------------|
| Single, with bath.....    | \$2.00 and \$2.50       |
| Double, with bath.....    | 3.00, \$3.50 and \$4.00 |
| Single, without bath..... | 1.50                    |
| Double, without bath..... | 2.00, 2.50, and 3.00    |

**TERMINAL HOTEL**

Location, Union Station. Rooms available, approximately 50.

|                           |               |
|---------------------------|---------------|
| Single, with bath.....    | \$2.00 and up |
| Double, with bath.....    | 3.50 and up   |
| Single, without bath..... | 1.50 and up   |
| Double, without bath..... | 2.50 and up   |

**WASHINGTON HOTEL**

Location, Kingshighway Boulevard and Washington Avenue.

|                   |                  |
|-------------------|------------------|
| Without bath..... | \$1.50 to \$2.00 |
| With bath.....    | 3.50 to 5.00     |

**MISSOURI ATHLETIC ASSOCIATION**

Club courtesies are extended. Can accommodate approximately from 75 to 80 gentlemen.

Rooms \$1.50 to \$3.00 per day with a few suites at \$4.00 to \$5.00.

Both shower and tub baths in the halls convenient to all bedrooms. Privilege of the natatorium, steam room, etc. Bath robes and slippers furnished to such guests as desire to make use of this privilege.

Privileges of other luncheon clubs and country clubs having golf courses for those desiring same.

**NOMINATIONS FOR OFFICERS**

The Committee on Nominations will welcome from members of the Institute suggestions for the offices which are to be filled by election in February, 1918, and which are as follows:

One officer, known as Director and President.

Two officers, known as Director and Vice-President.

Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N. B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.

District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.

District No. 4. Minnesota, Wisconsin, and Michigan.

District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.

District No. 6. California and Nevada.

District No. 7. Utah, Colorado, Arizona, and New Mexico.

District No. 8. Louisiana and Texas.

District No. 9. Other Southern States and District of Columbia.

District No. 10. Mexico.

District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President Philip N. Moore (not eligible for re-election).

Past President, William L. Saunders.

Vice-Presidents, Sidney J. Jennings, District 0; Joseph W. Richards, District 2.

Directors, Walter H. Aldridge, District 6; Robert W. Hunt, District 3; Hennen Jennings, District 9; George C. Stone, District 0; Samuel A. Taylor, District 2.

The Committee must make its final report to the Board of Directors early in October and members are therefore urged to send in their suggestions promptly. Communications should be sent to the Chairman of the Committee on Nominations, John V. N. Dorr, 17 Battery Place, New York, N. Y.

Every year the vote for officers is larger than the year before and it is hoped that the members will take an active interest in the nominations and will assist the Committee on Nominations with their suggestions and recommendations.

John V. N. Dorr, *Chairman*,  
Percy G. Beckett,  
J. W. Hutchinson,  
William H. Linney,  
Charles W. Merrill,  
E. Gybbon Spilsbury,  
Eli T. Conner.

### BURR A. ROBINSON

During the four years that Burr A. Robinson has held the position of Assistant Secretary of the Institute he has not only won the regard and admiration of the Directors and his associates at headquarters because of his unselfish devotion, efficiency and painstaking application, but he has formed a wide circle of friends among the members of the Institute. All will learn with regret that he has voluntarily severed his connection with the Institute in order to take a position with the United States Rubber Co., which offered such flattering prospects for the future that the Board of Directors, in spite of their disinclination to lose his services, did not feel justified in urging him to decline.

## NEW COLLECTIVE INDEX OF TRANSACTIONS

The Institute published a collective index of Volumes I to XXXV inclusive, which has been very extensively purchased and used by the members of the Institute. It enables one to look up any subject in the first thirty-five volumes and is useful to those who have the *Transactions*, as well as to those who have not, because it tells what subjects have been treated and enables one to look these matters up in his own library and then afterward either to procure the individual papers from the Institute at the price of \$1 each, or to procure Volumes, or to consult the *Transactions* at a nearby library, if such a one is available. It now seems desirable to publish another collective index, and the Editorial Department, upon the authority of the Board of Directors, has prepared an index of Volumes XXXVI to LV inclusive. This will be sold in the Institute's standard half morocco binding at a price of \$5 and in paper covers at a price of \$4. It will be ready in a few weeks. Those who already have the first index and purchase the second one, and those who purchase both, will have a complete index of the *Transactions* from Volumes I to LV inclusive. The Institute will be glad to receive advance orders for this index to insure prompt delivery upon publication.

## ALABAMA TECHNICAL ASSOCIATION

This Association is made up of Alabama members of the following technical societies: The American Society of Civil Engineers, American Institute of Mining Engineers, American Society of Mechanical Engineers, American Institute of Electrical Engineers, American Chemical Society and the American Institute of Architects. The association organized last winter for the purpose of maintaining an organization among the technical men of the State of Alabama, to stimulate the development of the natural resources of the State, to foster public interest in all things beneficial to the State, to advance the interest of the technical profession in the State and to encourage social intercourse among its members. On June 30, 1917, an excursion was made to the power house of the Alabama Power Co. at Lock 12 on the Coosa River. This is the first meeting held outside of Birmingham and is also the first meeting to which ladies were invited. About 85 members and guests attended and the excursion was voted a great success by all present.

## PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, JUNE 22, 1917

The President was authorized to appoint a committee comprising himself and five others, to draft an amendment to the Constitution for the purpose of bringing before the membership the suggested change in qualifications for members.

The President thereupon appointed the following Committee: Edwin Ludlow, Chairman, B. B. Thayer, J. W. Richards, G. C. Stone, C. F. Rand, and P. N. Moore.

The President was authorized to arrange with the Bureau of Mines for the payment of the necessary clerical assistance to copy the inventory of

personnel of mining engineers which had been collected by the Bureau of Mines in coöperation with the American Institute of Mining Engineers.

The Secretaries of Local Sections were invited to attend the 115th (St. Louis) meeting of the Institute, their traveling expenses, which shall include only railroad fare and Pullman, to be paid by the Institute.

The President reported that Julian Kennedy had asked to be excused from serving on the Nominating Committee. The President thereupon recommended Eli T. Conner for the Nominating Committee and he was appointed by the vote of the Board.

The Secretary reported the receipt of letters from some members of the Institute located in the Southeastern Missouri Lead District suggesting the postponement of the 115th meeting and it was

Resolved, that the plans for the 115th (St. Louis) meeting be continued unless the St. Louis Section withdraws the invitation.

The report of the Committee on Petroleum and Gas was presented in writing and was accepted in substance and referred to M. L. Requa for such revision in form as he may desire to make in view of suggestions which had been received from members of the Committee, after which the Secretary is authorized to transmit the recommendations to the New York Stock Exchange.

The Assistant Secretary, Burr A. Robinson, presented his resignation and this was accepted with regret. The Secretary recommended the appointment of a Managing Editor and of an assistant to the Treasurer and an assistant to the Secretary. Gar A. Roush was thereupon appointed Managing Editor for a period of two months. Chester A. Dick was appointed assistant to the Treasurer. Arthur L. Gresham was appointed assistant to the Secretary.

Upon the recommendation of the Committee on Membership ninety-seven candidates were elected. Three members were reinstated and four resignations were accepted. Two members who are engaged in active duty with the armies at the front had their dues suspended, and one member was suspended for non-payment of dues.

The report of the United Engineering Societies' Trustees was accepted.

The President reported that he had appointed A. L. Walker as a member of the Sub-Committee of General Engineering of the Advisory Commission to the Council of National Defense and this appointment was approved.

The President reported that he had appointed William Y. Westervelt as representative of this Institute on a Joint Committee of the U. S. Geological Survey, U. S. Bureau of Mines, the Association of State Geologists and the American Institute of Mining Engineers having to do with war minerals. This appointment was approved.

The President reported the resignation from the Executive Committee of L. D. Ricketts on account of ill health, and this resignation was accepted with regret.

J. E. Johnson, Jr., was appointed a member of the Executive Committee in place of L. D. Ricketts.

**MEMBERS OF THE INSTITUTE IN MILITARY SERVICE**

- AARONS, J. BOYD, Major.  
AGNEW, FRANK VANS.  
ALLEN, HERMAN H., Private, D Co., 8th Reserve Engineers.  
BANKS, H. R., Lieutenant, Canadian Engineers, B. E. F.  
BARBOUR, PERCY E., Deputy Supt. (Captain), New York State Troopers; Captain, 22d Regiment, N. Y. N. G.  
BARLING, H. B., 1st Lieutenant, Engineer Officers' Reserve Corps.  
BIGELOW, BRAXTON.  
BLOOMFIELD, E. C., Lieutenant.  
BORROW, F. K.  
BROOKS, ALFRED H., Captain, Engineer Officers' Reserve Corps.  
BROWNE, DE COURCY, 1st Lieutenant, Ordnance Dept., Fortress Monroe, Va.  
BULLOCK, S. C., Captain, Royal Engineers, British Army.  
CADMAN, JOHN, Liason Officer, B. E. F.  
CAETANI, GELASIO, 1st Lieutenant, Royal Italian Engineers.  
CAHEN, JAMES P., JR., Reserve Officer, U. S. Cavalry.  
CALDWELL, WILLIAM A.  
CHAMBOST, P. J. E. E.  
CHAPMAN, R. H., Major, Engineer Officers' Reserve Corps.  
CHURCH, JOHN L., Captain, Engineer Officers' Reserve Corps.  
DAVIS, A. W.  
DOUGALL, RALPH. Killed in action.  
DUNN, THEODORE S., 1st Lieutenant, Engineer Officers' Reserve Corps.  
DWIGHT, ARTHUR S., Major, Engineer Officers' Reserve Corps.  
EDWARDS-LECKIE, R. G., General.  
EMMERSON, ERIC ROY.  
EVANS, A. W., Lieutenant Colonel.  
FAIRBAIRN, GEORGE, Captain.  
FARRANT, JAMES C.  
FIDLAR, J. B., 2d Battery, Officers' Training Camp, Fort Logan, Ark.  
FORBES, CARROLL R., Captain, Engineer Officers' Reserve Corps.  
GATES, A. O., Lieutenant, U. S. Naval Reserve.  
GILCHRIST, A. D.  
GOODSPEED, G. E., JR., 1st Lieutenant, Engineer Officers' Reserve Corps.  
GORDON, ERNEST.  
GORMAN, THOMAS C.  
GRAVATT, C. MARSHALL, Aviation Corps, Columbus, Ohio.  
GRUNOW, W. R., 1st Lieutenant, Engineer Officers' Reserve Corps.  
GRUNSKY, CARL E., JR., Captain, Engineer Officers' Reserve Corps.  
HAGUE, WILLIAM, 1st Lieutenant, Engineer Officers' Reserve Corps.  
HALL, WILLIAM T.  
HARDINGE, HARLOWE, 1st Lieutenant, Signal Corps, U. S. R.  
HAYES, F. H., Captain and Adjutant, First Arizona Infantry.  
HEGGM, ALFRED G., Captain, Engineer Officers' Reserve Corps.  
HIBBERT, A., Lieutenant, 3d Canadian Tunnelling Corps, B. E. F.  
HINES, P. R., 1st Lieutenant, Engineer Officers' Reserve Corps.  
HOLLOWAY, W. S.  
HURD, RUKARD, Major, Engineer Officers' Reserve Corps.

- HUTCHISON, WILLIAM, Canadian Forestry Battalion, France.  
IRVING, J. D., Captain, Engineer Officers' Reserve Corps.  
KIRKPATRICK, G. H., Colonel.  
KNAPP, ARTHUR, Captain, Engineer Officers' Reserve Corps.  
KOPE, G. S.  
LACROIX, M. F., Captain, Engineer Officers' Reserve Corps.  
LAKELAND, WILLIAM J.  
LAVERY, VAUGHAN M., Lieutenant, 3d Tunnelling Corps, Canadian Engineers, B. E. F.  
LEONARD, W. H., Captain, Troop E, 1st Regiment, Colorado Cavalry.  
MACNUTT, C. H., Canadian Engineers' Training Depot., Crowborough, Sussex, England.  
MAGEE, JAMES F., 1st Regiment of Engineers, U. S. Expeditionary Force.  
MEWHIRTER, SYDNEY A., Designate Lieut., Corps of Engineers, U. S. Reserve.  
MILLARD Wm. J., Captain, 12th Engineers, U. S. R.  
MORGAN, G. H., Lieutenant, Engineer Officers' Reserve Corps.  
MUDD, SEELEY W., Major, Engineer Officers' Reserve Corps.  
NEWBERRY, R. W.  
OUGH, S. W.  
PARKS, HENRY M., Captain, Engineer Officers' Reserve Corps.  
PICKETT, Chas. E., B Battery, 5th Field Artillery, U. S. Army.  
PONTON, G. M., Lieutenant C. E. Explosives Dept., Imperial Munitions Board, Ottawa, Ont., Canada.  
PULLEN, ERNEST F.  
REECE, FRED B.  
REYNOLDS, GEORGE B.  
ROCKWELL, F. Y., Officers' Reserve Corps, Fort Myer, Va.  
ROSS, JAMES G., Captain, Discharged from active service.  
ROUDABUSH, M. A., 2d Lieutenant, Engineer Officers' Reserve Corps.  
SACKET, CHARLES T., Captain, Engineer Officers' Reserve Corps.  
SIBBALD, ALEXANDER.  
SMULLEN, C. Kenneth, 2d Lieutenant, Engineer Officers' Reserve Corps.  
STEHLLI, H. J., Captain, Engineer Officers' Reserve Corps.  
STEVEN, H. M., Lieutenant.  
VALLET, BENJAMIN W., Major, Battalion of Engineers, Michigan College of Mines.  
VAN NESS, W. WATERS, Captain, Royal Engineers, British Army.  
WATKINS, JOSEPH C., Captain, Engineer Officers' Reserve Corps.  
WEBB, TORREY H., Signal Service Reserve Corps.  
WEIGELL, HENRY S., Private, Artists' Rifles Platoon 9-C. Co., B. E. F.  
WEINBERG, GEORGE S., Major, Engineer Officers' Reserve Corps.  
WILKINSON, W. F., Captain.  
WORTH, JOHN G., Captain Quartermaster, Officers' Reserve Corps.  
WRAITH, C. R., Captain, Ordnance Dept., U. S. R.  
YOUNG, HORACE G.

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Request for Information.—We would appreciate information concerning the official position of members whose names are printed in the above list without title (who are now in active service at the front), and also concerning other members who are engaged in any military service.



## PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members)

The following is a partial list of members and guests who called at Institute headquarters during the period June 10, 1917, to July 10, 1917.

|  |   |
|--|---|
| Guy N. Bjorge, Boston, Mass.                             | H. H. Lauer, Allentown, Pa.             |
| A. H. Burroughs, Jr., Britannia Beach,<br>B. C., Canada. | E. C. Lee, Pittsburgh, Pa.              |
| Karl Eilers, New York, N. Y.                             | Montrose L. Lee, Port-au-Prince, Hayti. |
| H. S. Emlaw, Grand Haven, Mich.                          | Eugene McAuliffe, Paducah, Ky.          |
| W. R. Hamilton, San Francisco, Cal.                      | R. H. McMillen, Pittsburgh, Pa.         |
| Fred N. Hays, Sharon, Pa.                                | L. D. Moore, Providence, R. I.          |
| E. Hibbert, Nickelton, Ont., Canada.                     | Fred J. Pope, New Rochelle, N. Y.       |
| Charles Janin, San Francisco, Cal.                       | George S. Rice, Washington, D. C.       |
|  | W. G. Swart, Duluth, Minn.              |

**Milton A. Allen** has resigned his position with the Federal Lead Co. and accepted a position with the Arizona State Bureau of Mines, Tucson, Ariz.

**L. W. Bahney** has resigned as Professor at Yale University and accepted a position as metallurgical engineer with the Scovill Manufacturing Co., Waterbury, Conn.

**Guy N. Bjorge** has resigned his position as mine geologist with the Old Dominion Copper Mining and Smelting Co., Globe, Ariz., to take up general mine-examination work. His address will be 27 State St., Boston, Mass.

**Alfred H. Brooks**, formerly in charge of the Division of Alaskan Mineral Resources of the United States Geological Survey, has been appointed a captain in the Engineers' Reserve Corps and ordered to report for training. During his absence, George C. Martin will be in charge of Alaskan work.

**R. A. Bull** has accepted a position with the Duquesne Steel Foundry Co., Coraopolis, Pa.

**Frank R. Corwin**, formerly assistant superintendent of the International Smelting Co., has resigned to take charge as superintendent of the Consolidated Arizona Smelting Co.'s smelter at Humboldt, Ariz.

**Frank H. Crockard**, recently vice president of the Tennessee Coal, Iron and R. R. Co., has been elected president of the Nova Scotia Steel and Iron Co., Ltd. He succeeds Col. Thomas Cantley, who becomes chairman of the board of directors. Mr. Crockard will have charge of plans for a considerable extension of the company's operations.

**Ernest R. Cruther** has accepted a position as superintendent of the Electrolytic Zinc Plant of the Judge Mining & Smelting Co., Park City, Utah.

**Edwin R. Eaton** has resigned as manager of the Manganese Products Co., operating in Rye Valley, Virginia, and will take up other work in the south.

**J. R. Finlay**, who has been absent for a long time on professional business in the West, expects to return to New York presently. He has removed his office to 45 Cedar Street.

**Harold P. Ford** has accepted a position with the International Smelting Co., Tooele City, Utah.

**G. H. Gibbs**, formerly assistant chief engineer of the Coventry Ordnance Works, has resigned to become engineer for T. & I. Bradley & Sons, Ltd., Darlaston, Staffs, England.

**Rush M. Hess** has been appointed manager of the United Mines Co. of Arizona at Bouse, Yuma County.

**George R. Hilby** is now with the State Highway Commission, Sacramento, Cal.

**W. Spencer Hutchinson** has spent five months in Australia and New Caledonia. Among the many items of interest on the trip was a shipwreck in a cyclone, on one of the coral reefs off the shore of New Caledonia, from which Mr. Hutchinson was rescued by some natives in a whaleboat. He suffered no greater harm than the loss of part of his baggage.

**Reed W. Hyde** has accepted a position with the Dwight & Lloyd Sintering Co., Room 1500, 29 Broadway, New York, N. Y.

**Fayette A. Jones** has resigned from the Presidency of the New Mexico School of Mines and will be in Alaska for the remainder of the summer inspecting mining property.

**John S. Kennedy** has accepted a position with the Oriskany Ore & Iron Corpn., Lynchburg, Va.

**William H. Kobbé** has resigned as Petroleum Engineer from the Pierce Oil Corporation of Saint Louis and the Pierce Fordyce Oil Association and has entered private practice in Dallas, Texas, where he has a company in process of formation. On June 13, he was commissioned a First Lieutenant, Engineer Officers' Reserve Corps.

**F. E. Lathe**, head assayer of the Granby company's smelter at Grand Forks, has resigned to join the staff of the Chile Exploration Co. He was with the Granby company for nine years.

**Edward P. McCarty**, formerly professor of mining in the University of Minnesota, has established a general practice in mining engineering, with offices at 3615 Lyndale Avenue South, Minneapolis, Minn.

**H. Alfred Millard** announces the removal of his New York offices from 60 Wall St. to 2 Rector St.

**Henry S. Munroe**, emeritus professor of mining in Columbia University, has been awarded the medal given by the class of 1889 for the most distinguished piece of work done by a graduate of the Schools of Mines, Engineering and Chemistry

**Charles A. Randall** started operations at the Holguin Exploration Co.'s cyanide plant, Holguin, Cuba, on May 15. The plant was built under his supervision.

**H. M. Roche**, superintendent for the past five years of the Mt. Hope Mines, Empire Steel & Iron Co., has resigned and accepted the position of general superintendent of mines for the Wharton Steel Co., Wharton, N. J.

**Fred W. Sherman** has accepted a position as shift boss with the Arizona Copper Co., Metcalf, Ariz.

**H. I. Smith** has resigned his position as district mining engineer with the U. S. Bureau of Mines with headquarters at Urbana, Ill., to become assistant general superintendent of the Vandalia Coal Co., Terre Haute, Ind. Mr. Smith was with the Federal Bureau of Mines for seven years, during which time he was employed in conducting tests on mine roof supports, and explosives, and investigating mining practices in the central district of the United States.

**H. N. Stronck** has been appointed vice-president and managing engineer of the Illinois Engineering, Auditing and Public Relations Co., 122 S. Michigan Avenue, Chicago, this firm being successors to Benedict, Boyle & Stronck.

**J. E. Thropp, Jr.**, for some time superintendent of the Sheridan, Pa., blast furnace of the Berkshire Iron Works (owned by E. J. Lavino & Co., Philadelphia) and superintendent of the Marietta, Pa., blast furnace of the same interests, has resigned.

**J. B. Tyrrell**, of Toronto, has been appointed Canadian representative of the Consolidated Mines Selection Co., one of the strongest British mining companies, which has operated extensively in South Africa and now contemplates entering the northern Ontario field. Walter McDermott, the president of the company, began his mining career in Canada in connection with the old Silver Islet mine on the north shore of Lake Superior in 1872.

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## POSITIONS VACANT

No. 228. Slate mining company desires men for the following positions: Superintendent to take full charge of underground work, quarrying and milling. Assistant or night superintendent. Foreman for sinking a new shaft.

No. 229. Chief engineer to take charge of drafting room, on furnaces and combustion apparatus.

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## ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Mining engineer and executive. Member. Columbia graduate. Age, 32. Married. 9 years' responsible work in mine examination and development. Experience in United States, Canada, Mexico and South America. Now superintendent large operating mine. Personal interview in New York. Available in August. No. 367.

Member, graduate mining engineer, desires position as mine superintendent or assistant superintendent. Sixteen years' experience, in mill machine shop, shift boss, chief engineer and superintendent in copper and iron mines. Age, 37. Married. Location in town with good schools desired. Open for engagement. References. No. 365.

Member, married, age 35, at present employed, available about August 1st. Fine experience in Western U. S. A. Two years in Northern China. Specialty exploration and development of gold, silver, and copper. Good executive, excellent health and habits. Technical graduate. No. 368.

Member. Age, 24. Has had experience as assistant engineer, and in examining, surveying and measuring up coal mines. Desires position as surveyor. Will go anywhere. No. 369.

Member, A. B. and M. E. Age, 26. Has had experience as laborer, trackman, timberman, trammer, miner, surveyor, and efficiency engineer. Desires position as mining engineer, geologist or superintendent. No. 370.

Recent graduate, B. S. in mining engineering, desires position, preferably in a mill in Southwest United States. Has had experience, summers, as assistant to City Engineer, as material clerk, rod man and instrument man. No. 371.

Mine manager, member, technical graduate with many years experience, desires change. Now successfully managing producing mine in the Southwest. No. 372.

Member. 27 years old. Degree of E. M. Speaks Spanish. Desires position in mining dept. in U. S. or Canada. Has done surveying, has had underground and milling experience, and has been assistant to mine superintendent. No. 373.

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## LOCAL SECTION NEWS

### NEW YORK SECTION

J. E. Johnson, Jr., *Chairman*                      Edgar Rickard, *Vice Chairman*  
D. M. Liddell, *Secretary*, 7 Wall St., New York, N.Y.

John V. N. Dorr                      C. A. Bohn, *Treasurer*  
Lewis W. Francis

The annual meeting of the New York Section was held on Wednesday, June 6, with about 40 members in attendance.

The treasurer read a brief summary of the financial statement for the year, and the secretary's report was read. These reports were ordered placed on file together with the report of the auditors for the two previous years.

This was the first gathering since the death of the Chairman, Mr. David H. Browne, as the April meeting of the Section had been cancelled. Upon motion, a committee was appointed to draw up a resolution to be spread on the Minutes of the Section and a copy sent to Mr. Browne's family, expressing the loss and sympathy of the Section in Mr. Browne's death.

Officers for the ensuing year were unanimously elected, as shown above.

A recess was then taken for dinner.

Following the dinner we had as our principal speaker of the evening, Major George H. Putnam, who gave us a very interesting and stirring patriotic address comparing many of the conditions of the present time with those obtaining at the time of the Civil War, in which he served.

Captain Dulieux of the French Purchasing Commission, one of our members, was present and expressed a few words of greeting.

Mr. Drucker, a member of the Mining and Metallurgical Institute of London, also responded to the Chairman's request for some remarks from one lately returned from Great Britain.

In view of the situation in National affairs, it was decided to continue monthly meetings throughout the summer to take up problems in which our members would be qualified to be of assistance to the Government. The plans to be followed for these meetings are as follows:

1. It is suggested that we choose for discussion for each evening one specific military problem of which the Government desires a solution.

2. That we obtain the services of an expert in this subject to talk to us informally but in a technical way about this problem. That at the conclusion of his talk we have as general a discussion and as many questions asked bearing on this problem as possible, with the idea of eliminating the solutions that are known to be erroneous.

3. That we request the Institute to put a transcript of the talk and the discussions into the *Bulletin* and request the special attention of the entire membership to it, with the object of obtaining suggestions bearing on the problem from any member who feels able to offer them; these suggestions to be sent to the Secretary of the Institute and handled by a Committee or, if possible, a Commissioner paid for the purpose.

A. D. BEERS, *Secretary*, 1916-17.

## NEVADA SECTION

R. E. H. POMEROY, *Chairman*                      J. C. JONES, *Vice-Chairman*  
HENRY M. RIVES, *Secretary-Treasurer*, 210 Reno National Bank Bldg.,  
Reno, Nevada.

W. H. BLACKBURN

J. W. HUTCHINSON

EMMET D. BOYLE

JOHN G. KIRCHEN

FREDERICK BRADSHAW

C. B. LAKENAN

WHITMAN SYMMES

The second annual field meeting of the Nevada Section was held at Ely, McGill and Ruth, Nev., on June 22 and 23. The meeting was highly successful and the various visiting engineers and their ladies were loud in their praises of the hospitality of the engineers of the Ely District.

The first day's session was held at McGill, where about 45 members from various sections of the State were present. This was the technical session, which was opened with an address of welcome by Mr. C. V.

Jenkins, Business Manager of the Nevada Consolidated Copper Co. After a response by Chairman J. W. Hutchinson of the Nevada Section, the following papers prepared by members of the Nevada Consolidated staff were read and discussed:

Present Operation of Steam Shovel Mines, Nevada Consolidated Copper Company.—Robert Marsh, Jr., General Mine Superintendent.

Branch Raise System at the Ruth Mine, Nevada Consolidated Copper Company.—Walter S. Larsh, Underground Superintendent.

Ball Mill Practice at the Nevada Consolidated Company's Mill.—Geo. C. Riser, Concentrator Superintendent.

Coarse Crushing Practice at the Nevada Consolidated Copper Company's Mill.—Curtis N. Lindley, Jr., Engineer engaged in construction.

Handling and Roasting Fine Slime Concentrates at the Steptoe Plant of the Nevada Consolidated Copper Company, McGill, Nevada.—R. E. H. Pomeroy, Smelter Superintendent, and J. C. Kinnear, Assistant Smelter Superintendent.

Water Supply of the Nevada Consolidated Copper Company, Ely District, Nevada.—Lindsay Duncan, Mechanical Engineer.

Luncheon was served at the company mess, and during the afternoon an inspection was made of the smelter, concentrator and crushing plant of the company. In the evening a banquet was tendered to members and their ladies at the Steptoe Hotel in East Ely, after which the 54 guests participated in the dance which followed.

On the following morning the party proceeded to Ruth, where many went underground in the Nevada Consolidated mine, while others inspected the operation of the churn drills and the steam shovels in the great pit. After a luncheon served by the company, the business session of the Section was held, during which a paper was presented by Mr. Edward Steidle, Engineer in charge of Rescue Car No. 1, of the United States Bureau of Mines. An inspection was then made of the rescue car stationed temporarily at Ruth, and a demonstration by rescue teams of the Nevada Consolidated Copper Co. and Giroux Consolidated Mines Co. was given. The ladies, who on the previous day had been entertained at luncheon by Mrs. C. V. Jenkins of McGill, attended a luncheon given by Mrs. Walter Larsh of Ruth.

The Executive Committee elected for the ensuing year is given above.

In the evening, after the return from Ruth, members attended the Red Cross dance in Ely, in lieu of one that had been planned for the visiting engineers and their guests.

The Section adopted a resolution acknowledging the courtesy and hospitality of Mr. C. V. Jenkins, Mr. R. E. H. Pomeroy, Mr. J. C. Kinnear, Mr. Walter Larsh, Mr. Geo. C. Riser and the staff of the Nevada Consolidated Copper Co., whose activities and thoughtfulness had assured the success of the meeting.

Tonopah was designated for the next meeting.

HENRY M. RIVES, *Secretary*.

## UTAH SECTION

C. W. WHITLEY, *Chairman*      WILLIAM WRAITH, *Vice-Chairman*  
ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake  
City, Utah

CECIL FITCH

E. R. ZALINSKI

On May 24, the Utah Local Section held its first meeting of the year at the Utah Hotel, Salt Lake City. The meeting was primarily arranged in honor of Mr. Philip N. Moore, President of the Institute, who visited our Section on that date. The dinner was attended by about one hundred members and a few guests. An extremely interesting speech was made by Mr. Moore, which was most enthusiastically received. After the dinner, a paper entitled "The Enrichment and Segregation of Mill Tailings for Future Treatment" was presented by F. E. Marcy. This paper brought about a lively discussion which was chiefly carried on by R. C. Gemmell and Lafayette Hanchett.

Owing to the absence of the secretary of the section, W. C. Stimpson acted in this capacity both in assisting to arrange for the dinner and during the evening. Members of the Executive Committee met Mr. Moore upon his arrival at Salt Lake on the morning of May 24 and entertained him at the Country Club during the day. He was taken to the Utah Copper Mines at Bingham by R. C. Gemmell on May 25.

The members of the Section all expressed the hope that the visit of the President of the Institute to Local Sections may from this on be an annual affair.

ERNEST GAYFORD, *Secretary*.

## BOSTON SECTION

R. L. AGASSIZ, *Chairman*      JAMES G. CARLETON, *Vice-Chairman*  
E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology,  
Cambridge, Mass.

W. E. C. EUSTIS

G. A. PACKARD

The forty-fourth meeting of the Boston Section was held at the University Club on Monday evening, May 7, 1916. Eighteen members were present for the dinner and several more joined us later to listen to the reading of a paper on the Chilean Nitrate Industry by Allen H. Rogers of Boston. As this Chilean saltpeter furnishes the nitrogen for the manufacture of the bulk of the explosives of commerce, the paper was most timely, and it proved also to be intensely interesting and instructive. Starting with statistics of production, Mr. Rogers discussed the theories of origin, methods of prospecting and mining, mill treatment, living and labor conditions, transportation, costs, etc. The method of treatment appears crude and leaves probably an average of 8 per cent. of sodium nitrate in the tailings, so that it looks as if these might some day all be worked over again. Estimates of future supply vary from 20 to 200 years, with 50 years as a conservative estimate based on present methods of treatment. Improved methods of treatment would make it possible to treat a lower grade of material and doubtless very much prolong the life of the industry.

EDWARD E. BUGBEE, *Secretary*.

## COLUMBIA SECTION

W. H. LINNEY, *Chairman*OSCAR LACHMUND, *Vice-Chairman*LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154,  
Spokane, Wash.

STANLY A. EASTON

S. SHEDD

A meeting of the Columbia Section was held May 17 to 19, in connection with the International Mining Convention, as follows:

The usual luncheon of the Northwest Mining Association at the Spokane Hotel, May 17, was attended by officers and members of the Columbia Section and others of the American Institute of Mining Engineers, and after a brief address of welcome, President G. B. Dennis of the Association, who is also a member of the Institute, turned the meeting over to W. H. Linney, Chairman of the Columbia Section, who presided.

There were present about 100 persons. Representatives of the A. I. M. E. were President Philip N. Moore, Vice President C. W. Goodale, W. H. Linney, Chairman Columbia Section, L. K. Armstrong, Secretary and about 30 other members.

The meeting was also attended by Mayor Charles A. Fleming, J. L. Paine, President Spokane Chamber of Commerce, J. B. Fischen, member of Board of Managers, American Institute of Electrical Engineers, and Judge George Turner, each of whom spoke briefly in welcome to the visitors, who responded.

At 6:30 P. M., members of the Columbia Section and other members attended a banquet in honor of the visitors, which also was a business session, Chairman Linney presiding. President Moore addressed the members on the questionnaire given below.

The members expressed varying opinions in regard to the questions submitted, but the following summarizes what seemed to be the views of the majority of the members present.

Question 1. What do you think the American Institute of Mining Engineers can do to be of service to the Government in the present crisis?

Answer. We believe that all questions of service to the Government must be left to those of our Institute leaders that are most closely in touch with Government officials and their expert advisers.

Question 2. What does membership in the Institute mean to you?

Answer. Membership in the Institute means professional information, incentive and standing, together with a community of interest fostered by personal contact with other members chiefly through the medium of Section meetings.

Question 3. What should membership in the Institute mean?

Answer. Membership in the Institute should mean all that is stated in Answer No. 2, together with a personal pride in the advancement of the mining industry.

Question 4. What are your suggestions to improve the activities and standards of the Institute?

Answer. We have none except, perhaps, to urge increased Sectional activity (with special reference to Junior Members and Affiliated Student Societies) and to offer the opinions expressed in Answer 6 below.

Question 5. What are your criticisms of its management?

Answer. We have none to make.

Question 6. Do you think the time has come to consider seriously



making it a strictly professional organization, with no change in the status of the present members, but hereafter making more rigorous professional requirements necessary to membership?

Answer. In general, yes—but we are inclined to believe that professional lines too rigidly drawn might possibly work harm instead of good in many instances.

For example, in contemplating the "Suggested Requirements for Membership in the A. I. M. E. (Section 2)" we favor an age limit of 27 years instead of 30 years as proposed, and we believe that not only should considerable latitude be allowed the Committee on Membership in determining the qualifications of a particular candidate but also that the Committee should freely avail itself thereof on occasion.

An engineer, of whatever class, is a person skilled not only in the principles of his particular branch but more especially, perhaps, in its practices. In other words, he is one who makes physical rather than oral application of talents whetted by culture and training.

President Moore thanked the members and said that their opinions would be duly considered in the final analysis of the matter. He then gave a résumé of the library service system, where desired data may be had at a low cost.

F. A. Ross offered the following resolution which was seconded by J. V. Richards and, being submitted, was carried unanimously, the secretary being ordered to make suitable record and publicity of same, after which the meeting adjourned to enable the members of the party to en-train for Nelson, B. C., where a joint meeting of Columbia Section, A. I. M. E. and Western Branch C. M. I. was to be held in connection with the International Mining Convention.

### *Resolution*

WHEREAS, on May 5, 1915, the members of Columbia Section, A. I. M. E., were honored and greatly benefited by a visit from Secretary Stoughton and Professor Joseph W. Richards, and

WHEREAS, on May 17, 1917, they were again privileged to honor as guests President Philip N. Moore and Vice-President Charles W. Goodale, and

WHEREAS, all members of this Section are agreed that personal contact with high officials of the Institute makes for a better understanding of the plans and procedure of the Board of Directors and the great work of the Institute as a whole, thereby binding the West more closely to the East, now, therefore, be it

RESOLVED that Columbia Section, A. I. M. E., unanimously urges the Board of Directors to encourage such visits in future, particularly to those Sections lying farthest from New York, and to endeavor to find a means of making them annual events at the expense of the Institute. And be it further

RESOLVED that this resolution be spread upon the minutes of the Section and that a copy thereof be submitted to the Board of Directors.

### *International Mining Convention*

At 10 p. m. four sleepers left the Union Station over the Spokane International division, Canadian Pacific Railway, for Nelson, B. C., with 100 persons aboard.

On arrival, after registration at the several hotels, the delegates attended the opening ceremonies at Eagle Hall, where addresses of welcome were made by J. J. Malone, President of the Nelson, Slocan and Eastern B. C. Mining Men's Club, Mayor J. E. Annable, City of Nelson, and others.

A response was rendered by C. A. Fleming, Mayor of Spokane.

Then followed the technical and social sessions, as outlined in the July *Bulletin*, p. xxxv.

G. B. Dennis, President of the Northwest Mining Association, did more than his share in bringing about the success of the International Mining Convention. He and F. C. Bailey, Secretary of the Northwest Mining Association, gave freely of their time and effort to bring about this noteworthy meeting.

L. K. ARMSTRONG, *Secretary*.

### FORTHCOMING MEETINGS OF SOCIETIES

| Organisation                                    | Place               | Date<br>1917 |
|---|---------------------|--------------|
| American Society of Sanitary Engineers.....     | Grand Rapids, Mich. | Aug. 1-3     |
| National Association of Stationary Engineers... | Evansville, Ind.    | Sept. 10-15  |
| National Safety Council.....                    | New York City.      | Sept. 10-15  |
| National Exposition of Safety and Sanitation..  | New York City       | Sept. 10-15  |
| American Iron, Steel & Electrical Engineers.... | Philadelphia, Pa.   | Sept. 10-14  |
| American Institute of Metals.....               | Boston, Mass.       | Sept. 24-29  |
| American Foundrymen's Association.....          | Boston, Mass.       | Sept. 24-29  |
| Chemical Industries, Third National Exhibition. | New York City       | Sept. 24-29  |
| American Electrochemical Society.....           | Pittsburgh, Pa.     | Oct. 3-6     |
| American Institute of Mining Engineers.....     | St. Louis, Mo.      | Oct. 8-13    |
| American Gas Institute.....                     | Washington, D. C.   | Oct. 16-19   |
| American Institute of Architects.....           | Philadelphia, Pa.   | Dec. 26-29   |
| American Society of Mechanical Engineers....    | New York City       | Dec. 26-29   |
| American Mining Congress State Chapter.....     | Phoenix, Ariz.      | Dec. 26-29   |

**LIBRARY**

AMERICAN SOCIETY OF CIVIL ENGINEERS.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.

AMERICAN SOCIETY OF MECHANICAL ENGINEERS.

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

HARRISON W. CRAVER, Director

The Library of the above-named Societies is open from 9 A. M. to 10 P. M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

**LIBRARY ACCESSIONS****PARTIAL LIST CLASSIFIED BY SUBJECTS****Mining and Metallurgy**

COMITÉ CENTRAL DES HOUILLÈRES DE FRANCE. Codes Miniers—Russie. Paris, 1895.

EFFICIENT PURCHASE AND UTILIZATION OF MINE SUPPLIES. By H. N. Stronck and J. R. Billyard. New York, 1917.

MONTHLY STATEMENT OF COAL MINE FATALITIES IN THE UNITED STATES, MARCH, 1917. List of permissible explosives, lamps, and motors tested prior to April 30, 1917. Washington, 1917.

TRANSVAAL CHAMBER OF MINES. Address delivered by E. A. Wallers at the Annual Meeting of Members, March 26, 1917. Johannesburg, 1917. (Gift of Transvaal Chamber of Mines.)

UNITED MEXICAN MINING ASSOCIATION. Report of the Court of Directors, dated June 13, 1827. London, 1827.

WORKS OF DMITRI KONSTANTINOVICH CHERNOFF. Published by the Russian Metallurgical Society. Petrograd, 1915. (Gift of D. K. Chernoff.)

**Geology and Mineral Resources**

BYRON OIL AND GAS FIELD, Big Horn County. (Wyoming State Geologist's Office. Bulletin No. 14.) Cheyenne, 1917.

CANADA. Mines Department. Report on the building and ornamental stones of Canada. Vol. IV. Ottawa, 1916.

- GEOLOGICAL AND MINING SOCIETY OF AMERICAN UNIVERSITIES YEAR BOOK AND DIRECTORY, 1916. California, Stanford Univ., 1916. (Gift of Society.)
- GLADSTONE MINERAL DISTRICT. (Tasmania. Geological Survey. Bulletin 25.) Tasmania, 1916.
- OHIO. INDUSTRIAL COMMISSION. Statistics of Mines and Quarries in Ohio. 1913, 1914. (Gift of George Komar-of-Skye.)
- INDIANA, INDUSTRIAL BOARD. Annual Reports of the Department of Mines & Mining. 1914-15. (Gift of George Komar-of-Skye.)
- MICHIGAN, Geological map of. 1916.
- MINERAL PRODUCTION OF CANADA DURING THE CALENDAR YEAR, 1915. Ottawa, 1914. (Gift of George Komar-of-Skye.)
- NEW ZEALAND OFFICIAL YEAR BOOK, 1916. Wellington, 1916. (Gift of New Zealand, Registrar General.)
- OIL AND PETROLEUM MANUAL, 1917. By Walter R. Skinner. London, 1917.
- OIL FIELD DEVELOPMENT AND PETROLEUM MINING. By A. B. Thompson. New York, 1916.
- PENNSYLVANIA. DEPARTMENT OF MINES. Report. 1914. Vols. 1-2. (Gift of George Komar-of-Skye.)
- PETROGRAD. University of Commerce and Manufacture. Report of the Department of Mines. 1911. (Gift of George Komar-of-Skye.)
- THE MOLYBDENUM INDUSTRY IN NEW SOUTH WALES. (New South Wales. Department of Mines. Mineral Resources No. 24.) Sydney, 1916.
- WASHINGTON. STATE INSPECTOR OF COAL MINES. Report biennial period ending Dec. 31, 1916. Olympia, 1917. (Gift of Washington State Inspector of Coal Mines.)

### Chemistry and Testing

- CHEMICAL ANALYSIS OF LEAD AND ITS COMPOUNDS. By John A. Schaeffer and Bernard S. White. Joplin, 1912. (Gift of Eagle Picher Lead Company.)
- CHEMISTRY OF MATERIALS OF THE MACHINE AND BUILDING INDUSTRIES. By Robt. B. Leighou. New York, 1917.
- TEXT BOOK OF THERMO-CHEMISTRY AND THERMODYNAMICS. By Otto Sackur, translated and revised by G. E. Gibson. New York-London, 1917.
- TREATISE ON QUANTITATIVE INORGANIC ANALYSIS. By J. W. Mellor. London, 1913.

### General

- AMERICAN ELECTROCHEMICAL SOCIETY. Transactions. Vol. XXX, 1916. South Bethlehem, 1917.
- AMERICAN GAS INSTITUTE. Proceedings. Volume XI (Pt. 1-2). New York, 1917.
- GERMAN ENGLISH DICTIONARY FOR CHEMISTS. By A. M. Patterson. New York, 1917.
- NEW INTERNATIONAL YEAR BOOK, 1916. New York, 1917.
- PARALLEL TABLES OF SLOPES AND RISES. By C. K. Smoley. New York, 1917.
- PIONEERS OF SCIENCE. By Sir Oliver Lodge. London, 1910. (Gift of Frank Gilliland.)
- ROYAL SOCIETY OF VICTORIA. Transactions. Vol. VI, 1914. Melbourne, 1914.
- STEAM TURBINES. Ed. 3. By J. A. Moyer. New York, 1917.
- STRESSES IN STRUCTURAL STEEL ANGLES WITH SPECIAL TABLES. By L. A. WATERBURY. New York, 1917.
- THE THEORY OF MACHINES. By Robt. W. Angus. Ed. 2. New York, 1917.
- TOPICOS TECNICOS. By Octavio A. Acevedo. Santo Domingo, R. D., 1917. (Gift of author.)

### Company Reports

- AMERICAN SMELTING AND REFINING COMPANY. Annual Report 18th, 1916. (Gift of Judd Stewart.)

### Trade Catalogs

- ALLIS-CHALMERS MANUFACTURING Co. Milwaukee, Wis. Book No. 1632-A. Centrifugal Pumps and Centrifugal Pumping Units. Ed. 3, July, 1916; Ed. 4, May, 1917.
- GENERAL ELECTRIC Co. Schenectady, N. Y. Bulletin No. 44406-A. GE-247 Ventilated commutating-pole railway motor. March, 1917.

- 48900. CR-9510 Automatic solenoid brakes and CR-9500 solenoids. April, 1917.
- CATALOG OF ELECTRIC HOUSEHOLD APPLIANCES. (Y-986.)  
 INGERSOLL-RAND Co. New York, N. Y. Form No. 8507. "Little David" pneumatic drills. Feb., 1917.
- SCHIEREN, CHAS. A., Co. New York, N. Y. Leather Belting. 1917.

### Book Notices

Unless otherwise specified, books in the following list have been donated by the publishers. The Institute does not assume responsibility for any statements made. They are taken either from the preface or the text of the book.

**COMPRESSED AIR PRACTICE IN MINING.** By David Penman, M. Inst. M. E., Phila., J. B. Lippincott Co.; Lond., Charles Griffin & Co., Ltd., 1917. 221 pp., 113 illus., 5 × 8 in., cloth.

Text-book for use in mining schools, but also useful, it is hoped, to those in practice. Discusses air-compressors, methods of transmitting power and the use for compressed air for coal cutting, conveyors, drills, etc. A comparison of the advantages of compressed air and other means of transmitting power in mines is given.

**COMPRESSED AIR FOR THE METAL WORKER.** By Charles A. Hirschberg. N. Y., The Clark Book Co., 1917. 321 pp., 294 illus., 5 × 8 in., cloth, \$3.

Describes the various purposes for which compressed air is used in power plants, foundries, machine shops, forge shops, etc. The various types of tools and machines are illustrated, as well as the commercial forms of air compressors. A compendium of present-day method of utilizing compressed air, confined entirely to practice and omitting theory.

**HIGH SPEED INTERNAL COMBUSTION ENGINES.** By Arthur W. Judge. N. Y., The Macmillan Co.; Lond., Whittaker & Co., 1916. 9 + 350 pp., 217 illus., 6 × 9 in., cloth, \$5.50.

The author has collected and classified the available information, and presents it as briefly as possible. The work discusses the theory of high speed internal combustion engines, and the experimental results which have been obtained. Contents: The Thermodynamics of the Internal Combustion Engine; The Conditions Occurring in Actual Engines; Pressures and Temperatures in Internal Combustion Engines; Indicators and Indicator Diagrams; the Mechanics of the High Speed Internal Combustion Engine; Engine Balance.

**STEAM TURBINES.** A Practical Work on the Development, Advantages and Disadvantages of the Steam Turbine; The Design, Selection, Operation, and Maintenance of Steam Turbine and Turbo-Generator Plants. By Walter S. Leland. Chicago, American Technical Society, 1917. 137 pp., 116 illus., 6 × 8 in., cloth, \$1.

Intended for those less interested in the finer points of steam turbine theory than in the results accomplished and the way in which they have been secured by the successful builders. Over one-half of the book is devoted to descriptions of various commercial turbines.

**THE POWER KINK BOOK.** Novel Ideas and Simple Devices for Meeting Emergencies in the Power Plant, Compiled from the Regular Issues of "Power." N. Y., Power, 1917. 146 pp., 147 illus., 6 × 9 in., boards, \$1.75.

A collection of methods devised by power plant engine runners and machinists for making repairs, overcoming difficulties and preventing accidents. The methods are classified into appropriate groups and clearly illustrated by numerous drawings and sketches.

**APPLIED MECHANICS.** By Alfred Poorman. N. Y., McGraw-Hill Book Co., Inc.; Lond., Hill Publishing Co., Ltd., 1917. 11 + 244 pp., 321 illus., 6 × 9 in., cloth, \$2.

A text-book for undergraduate courses in engineering schools, in which the basic principles have been developed in a way which the average student can follow easily and illustrations showing clearly the application of such principles to the solution of engineering problems have been provided. Extended use has been made of the graphic method of solution.

**LAWS OF PHYSICAL SCIENCE.** A Reference Book. By Edwin F. Northrup. Phila. & Lond., J. B. Lippincott Co., 1917. 7 + 210 pp., 5 × 8 in., leather, \$2.

A summary of the general propositions underlying physical science and engineering, classified for convenience in consultation. Each law is accompanied by one or more references to more extended information. A bibliography is included.

**HANDBOOK OF CHEMISTRY AND PHYSICS.** A Ready-Reference Pocket Book of Chemical and Physical Data. 5th ed. Cleveland, The Chemical Rubber Co., 1917. 414 pp., 4 × 7 in., cloth, \$2.

The present edition of this convenient reference book has been carefully revised, many new tables have been added and the index enlarged.

**THE LIFE OF ROBERT HARE; an American Chemist (1781-1858).** By Robert Fahs Smith, Provost of the University of Pennsylvania. Phila. & Lond., J. B. Lippincott Co., 1917. 508 pp., 1 por., 4 illus., 6 × 9 in., cloth, \$5.

Assembles the labors of Robert Hare in such a form as to acquaint students of chemistry with him, and to show his title to an exalted place in the history of chemistry in this country. Told largely by hitherto unpublished letters and documents collected from forgotten journals and pamphlets.

**OUTLINES OF INDUSTRIAL CHEMISTRY.** A Text-Book for Students. By Frank Hall Thorp, with assistance in revision from Warren K. Lewis. 3d ed. rev. & enl. N. Y., The Macmillan Co., 1916. 25 + 665 pp., 137 illus., 6 × 9 in., cloth, \$3.75.

The present edition of this well-known treatise has been thoroughly revised, many sections having been rewritten and much obsolete matter replaced by new material. The modern concepts and theories of chemistry have been introduced whenever these promised to make clearer the phenomena involved.

**A TEXT-BOOK OF INORGANIC CHEMISTRY.** Edited by J. Newton Friend. Vol. VIII. The Halogens and Their Allies. By Geoffrey Martin and E. A. Dancaaster. Lond. Charles Griffin & Co., Ltd., 1915. 18 + 337 pp., 30 illus., 1 tab., 6 × 9 in., cloth, \$3. (Gift of J. B. Lippincott.)

A concise general account of the chief chemical and physical properties of fluorine, chlorine, bromine, iodine, and manganese, and their compounds. Describes the most important manufacturing operations briefly. The work does not attempt to be exhaustive, but is provided with very numerous references to original publications on the various phases of the subjects dealt with.

**ELEMENTARY CHEMICAL MICROSCOPY.** By Emile Monnin Chamot. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1916. 13 + 410 pp., 147 illus., 6 × 9 in. cloth, \$7.

The author believes that the American chemist has failed to appreciate the benefits to be obtained by the application of chemical microscopic methods in the industries and in research. This book is written to instruct him in the manipulation of the microscope and in the great variety of problems which may be solved by its use, and to call attention to the fields in which it is especially useful.

**CHEMICAL TESTS FOR MINERALS.** By Arthur J. Burdick. Beaumont, Cal., The Gateway Publishing Co., 1917. 93 pp., 5 × 8 in., cloth, \$1.25.

Handbook of simple qualitative tests, intended to enable prospectors without chemical training to identify the various rocks and ores met with in the field.

**ANNUAL CHEMICAL DIRECTORY OF THE UNITED STATES.** B. F. Lovelace, Editor. Baltimore, Williams & Wilkins Co. (copyright, 1917). 305 pp., 6 × 9 in., cloth, \$5.

First issue. Includes American manufacturers of and dealers in chemical apparatus and equipment; and professional chemical firms and laboratories. Lists of colleges offering instruction in chemistry, experiment stations, technical and scientific societies are given, also of Federal and State officials of dairying, foods, drugs, etc. Bibliographies of American and foreign journals and important books of the year, with a concise review of new happenings, devices, methods and appliances conclude the book.

**ELECTRIC RAILWAY TRANSPORTATION.** By Henry W. Blake and Walter Jackson. N. Y., The McGraw-Hill Book Co., Inc.; London., Hill Publishing Co., Ltd., 1917. 7 + 487 pp., 120 illus., 6 × 9 in., cloth, \$5.

The first book, the authors state, devoted to transportation methods and practice. Largely compiled to put in convenient form matter which has appeared in the transactions of the American Electric Railway Transportation and Traffic Association and the electric railway periodicals. Contents: Organization and Definitions; Adjustment of Service to Traffic; Accelerating Traffic Movement along the Line; Accelerat-

ing Traffic Movement on the Car; Car Types in Relation to Traffic; City Timetables—Preliminaries; Interurban Schedules and Dispatching; Fares; Fare Collection Practices and Devices; Public Relations; Promotion of Passenger Traffic; Traffic Signs for Cars, Station and Road-Information for the Public; Competition; Freight and Express Business; Selection and Training of Men; Wages and Wage Agreements; Welfare Work; Discipline of Trainmen; Forms of Extra Pay.

**STORAGE BATTERIES SIMPLIFIED.** Operating Principles—Care and Industrial Applications, a Complete, Non-technical but Authoritative Treatise Discussing the Development of the Modern Storage Battery, Outlining the Basic Operation of the Leading Types, also the Methods of Construction, Charging, Maintenance and Repair. All Practical Applications of Commercial Batteries are Shown and Described. By Victor W. Pagé. 208 pp., 89 illus., 5 × 8 in., cloth, \$1.50.

Special Instructions for Care and Repair of Automobile Batteries and a Glossary of Terms are included.

**THE PANAMA CANAL AND COMMERCE.** By Emory R. Johnson. N. Y. & Lond., D. Appleton & Co., 1916. 295 pp., 8 illus., 7 maps, 5 × 7 in., cloth, \$2.

Explains the reasons for the building of the canal and discusses the use of the waterway by the commerce and shipping of the United States and other countries. Intended for those engaged in shipping and other students of the canal in relation to commerce. A companion volume to Gorgas' "Sanitation in Panama" and Sibert and Stevens, "The Construction of the Canal."

**IRRIGATION WORKS CONSTRUCTED BY THE UNITED STATES GOVERNMENT.** By Arthur Powell Davis. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1917. 16 + 413 pp., 128 illus., 6 × 9 in., cloth, \$4.50.

Contains engineering descriptions, with the necessary illustrations, of the various projects undertaken by the Reclamation Service. The projects described are the Salt River, the Yuma, the Orland, the Grand Valley, the Uncompahgre, the Boise, the Minidoka, the Huntley, the Lower Yellowstone, the North Platte, the Truckee-Carson, the Carlsbad, the Hondo, the Rio Grande, the Umatilla, the Klamath, the Belle Fourche, the Strawberry Valley, the Okanogan, the Yakima and the Shoshone. Intended for engineers and those interested in the development of arid lands.

**BUILDING SUPERINTENDENCE FOR STEEL STRUCTURES.** A Practical Work on the Duties of a Building Superintendent for Steel-Frame Buildings and the Proper Methods of Handling the Materials and Construction. By Edgar S. Belden. Chic., American Technical Society, 1917. 95 pp., 24 illus., 2 pl., 5 × 8 in., cloth, \$1.

Concise practical discussion of the problems which confront the superintendent of structural steel construction, and of the proper methods of meeting them.

**ELLIOTT'S WEIGHTS OF STEEL.** For Engineers, Architects, Contractors, Builders, Steel Manufacturers, and all Users of Rolled Steel. Computed by Thomas J. Elliott. Cleveland, The Penton Publishing Co., 1916. 662 pp., 6 × 9 in., leather, \$20.

Given the weight of a lineal foot of any section of rolled steel, this book of tables enables one to find the weight of a single piece of any length without performing the operation of multiplication.

**PLAIN AND ORNAMENTAL FORGING.** By Ernst Schwarzkopf. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1916. 10 + 267 pp., 228 illus., 5 × 8 in., \$1.50.

Written to provide a really practical treatise on the theory and practice of art metal and blacksmith work, suitable for use as a text-book for beginners. Simple and detailed drawings illustrating each important operation are provided, together with full explanations. Intended especially for self-instruction. The author has had many years experience as a blacksmith and an instructor in forge work.

**STRESSES IN STRUCTURES.** By A. H. Heller, rev. by Clyde T. Morris. 3d ed. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1916. 13 + 374 pp., 230 illus., 6 × 9 in., cloth, \$2.75.

Author has attempted to provide a book which, without being an exhaustive scientific treatise on stresses, would be a suitable text-book for students and also a concise reference book for engineers. In the present edition various explanations have been expanded, numerical illustrative examples have been added and new material introduced where the reviser has found it necessary.

**ARITHMETIC FOR ENGINEERS;** Including Simple Algebra, Mensuration, Logarithms, Graphs, and the Slide Rule. By Charles B. Clapham. (The D. U. Technical Series), N. Y., E. P. Dutton & Co., (preface, 1916) 436 pp., 149 illus., 6 × 9 in., cloth, \$3.

The author's object is to treat the subject with sufficient detail and enough engineering application to provide a truly practical treatise, omitting all subjects which have only academic interest.

**THE TAYLOR SYSTEM OF SCIENTIFIC MANAGEMENT.** By C. Bertrand Thompson. N. Y., Chic., Lond., A. W. Shaw Co., (copyright 1917) 175 pp., 22 illus., 5 diagrams, 9 × 11 in., flexible cloth, \$10.

Intended to give, in addition to the history and theory of the system, enough development and operation to enable the factory manager to visualize the system in some detail, to distinguish clearly between it and other systems, and to understand its principles and mechanisms as found in actual practice. Based on personal investigations of all the installations of the system between Maine, Maryland and Chicago. Contains a bibliography of the important publications on the system.

**THE ESSENTIALS OF AMERICAN TIMBER LAW.** By J. P. Kinney, N. Y.. John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1917. 19 + 279 + 10 pp., 6 × 9 in., cloth, \$3.

A presentation of the existing law regarding trees and their products as property, with such observations and references to historical development as are considered necessary to an understanding of the reasons for the law. Citations to the sections of the compiled or session laws of the different states enable the reader to find the various statutes in full. Especial prominence is given to interpretations of the law by the courts.

**WORKMEN'S COMPENSATION LAW.** Personal Injury by Accident Arising Out of and in the Course of the Employment. By P. Tecumseh Sherman. N. Y., Workmen's Compensation Publicity Bureau (copyright, 1916). 67 pp., 6 × 9 in., paper, \$2.

A compilation of the decisions construing the British law on the subject, with abbreviated summaries of the relevant portions of the French and German laws. These precedents will be useful, the author believes, in defining the meaning of "accidents due to risk of work" as used in the American statutes.

**FIRE PREVENTION AND PROTECTION.** A Compilation of Insurance Regulations Covering Modern Restrictions on Hazards and Suggested Improvements in Building Construction and Fire Prevention and Extinguishment. 3d ed. Completely rev. by A. C. Hutson, Fire Protection Engineer. N. Y. & Ohio. The Spectator Co., 1916. 7 + 777 pp., 126 illus., 3 pl., 5 × 7 in., leather, \$4.25.

Written especially for merchants, manufacturers and underwriters, as a succinct presentation of the knowledge necessary to attain as complete protection as possible from fire. Covers all the suggested regulations of the National Board of Fire Underwriters and allied organizations.

**MILITARY SKETCHING AND MAP READING.** By Capt. Loren C. Grieves. Wash., U. S. Infantry Assoc., 1917. 95 pp., 32 illus., 6 × 9 in., cloth, \$1.

A text-book intended for officers of the National Guard, candidates for commissions in the Army and the Reserve Officers' Training Corps and for educational institutions. Meets the provisions prescribed by the War Department.

**TOPOGRAPHICAL DRAWING.** By Edwin R. Stuart, N. Y. McGraw-Hill Book Co. Inc.; Lond., Hill Publishing Co., Ltd., 1917. 126 pp., 54 illus., 1 map, 6 × 9 in., cloth, \$2.

Intended to furnish a satisfactory standard of practice in topographical drawing which will combine good execution with economy of time. Contents: Introductory; Map Projection; Instruments and Drawing Materials; Plotting; Special Methods in Free-Hand Drawing; Practice in Topographical Drawing; Lettering; Conventional Signs; Map Drawing.

**HEATONS' ANNUAL.** The Commercial Handbook of Canada and Boards of Trade Register 13th Year 1917. Toronto, Heaton's Agency, 1917. 518 pp., 5 × 7 in., cloth, \$1.25.

Collects in one volume the information of value to merchants and manufacturers. Includes lists of government officials, customs brokers, banking towns, registration offices, shipping directions, etc. The customs tariff and a digest of customs law and



regulation are given. The work also contains much general information on the resources of the country, a gazetteer of commercial towns and an economic bibliography of governmental reports, together with the usual tables.

**MAGNETIC AND OTHER PROPERTIES OF IRON-ALUMINUM ALLOYS. MELTED IN VACUO.**

By Trygve D. Yensen and Walter A. Gatward. Bulletin 95, Engineering Experiment Station, University of Illinois, 1917, 54 pp., 9 × 6 in.

The Engineering Experiment Station of the University of Illinois has just completed another series of experiments to determine the magnetic and allied properties of alloys of iron and other metals. In 1915 experiments with iron-silicon alloys disclosed some remarkable properties which make it superior to any other material for use in certain electro-magnetic machinery. These experiments were followed by others dealing with iron-aluminum alloys melted in vacuo, which have shown that aluminum like silicon greatly improves the magnetic properties of the metal, and also that aluminum imparts to the metal a greater toughness than silicon.

**SCHEMATIC OUTLINE OF THE METALLURGY OF IRON AND STEEL.**

Professor Frank E. Sanborn, Director of the Industrial Arts Department, Ohio State University, has prepared a flow sheet or outline of the Metallurgy of Iron and Steel for his own use in teaching the subject. A number of interesting features are contained in this chart, of which duplicates may be obtained at a small price by application to Professor Sanborn.

## MEMBERSHIP

## NEW MEMBERS

The following list comprises the names of those persons who became members during the period of June 10, 1917, to July 10, 1917.

- BANKS, LEON MAXWELL . . . . . Agent & Supt., Grant Leasing Co., Metcalf, Ariz.  
 BEAUCHAMP, HENRY CLAY . . . . . Mine Operator, Sisson, Siskiyou Co., Cal.  
 BOCKING, CHARLES, Asst. Mgr., Butte & Superior Mining Co., Rialto Bldg.,  
 Butte, Mont.  
 BEYER, HERMAN E., Chief Engr., Condenser Dept., Ingersoll-Rand Co., 11 Broadway,  
 New York, N. Y.  
 BLACK, HERBERT F. . . . . 1812 Oliver Bldg., Pittsburgh, Pa.  
 BRENNAN, PATRICK J., Gen'l. Supt., Elkhorn Division, Consolidation Coal Co.,  
 Jenkins, Ky.  
 BUTNER, DANIEL W., Chem. & Supt., Underground Min., Spanish American  
 Iron Co., Daiquiri, Santiago de Cuba, Cuba.  
 CARLTON, DAVE P. . . . . Nevada-Douglas Cons. Copper Co., Ludwig, Nev.  
 DE AROZENA, JOE . . . . . Min. Engr., 321 Preusser St., San Angelo, Texas.  
 DICKIE, CHARLES HERBERT, Pres., Georgia River Min. Co.,  
 Duncan, Vancouver Island, B. C., Canada.  
 DOBBINS, WALTER . . . . . Mill Foreman, Chino Copper Co., Hurley, New Mexico.  
 DRURY, CHARLES W., Asst. Professor of Met., Queen's University,  
 Kingston, Ont., Canada.  
 ENGLISH, EDWARD . . . . . Mine Sampler, Homestake Mining Co., Lead, So. Dak.  
 FIELD, THOROLD F., Min. Engr., Congdon Estate, 807 Lonsdale Bldg., Duluth, Minn.  
 FREY, FREDERICK C., Gen'l. Mgr., Mijnbouw Maatschappij, Redjang Lebong,  
 Lebong, Donok via Benkelen, Sumatra.  
 GIRVIN, WILLARD S. . . . . Asst. Met., The American Brass Co., Waterbury, Conn.  
 GRUVER, ESDRAS L. . . . . 25 Broad Street, New York, N. Y.  
 GUNDERSON, GERHARD WILLIAM . . . Min. Captain, Inland Steel Co., Crosby, Minn.  
 GUPTA, DHIRENDRA CHANDRA, Genl. Foreman, By-Product Coke Ovens,  
 Tata Iron & Steel Co., Sakchi, India.  
 HARRINGTON, R. F., Chem. & Met., Hunt-Spiller Mfg. Corp.,  
 383 Dorchester Ave., Boston, Mass.  
 HARRIS, W. G., Mgr., W. G. Harris & Co., Lessees, 726 Old National Bank Bldg.,  
 Spokane, Wash.  
 HENDERSON, GEORGE M. . . . . Min. Engr., Box 3, Weston, Mass.  
 HENRY, EARL CHARLES, Chief Engr., Witherbee-Sherman & Co., Mineville, N. Y.  
 HOGG, WILLIAM LANGLANDS, Min. Engr., Genl. Mgr., Crimora Manganese Corp.,  
 Crimora, Va.  
 HOSLER, RUSH N., Supervisor of Coal Schedule Rating, P. O. Box 472, Harrisburg, Pa.  
 HUGHES, VICTOR HARMON, Cons. Geol., Valerius, McNutt & Hughes,  
 329 Mayo Bldg., Tulsa, Okla.  
 JANNEY, FRANK G. . . . . Genl. Supt. of Mills, Utah Copper Co., Garfield, Utah.  
 JOHNSON, LAFAYETTE G. . . . . Mining Supt., Federal Lead Co., Flat River, Mo.  
 JORGENSEN, ERLING LOSSIUS, Met. Engr., Chile Exploration Co.,  
 Chuquicamata, Chile, South America.  
 KENNARD, EDWIN H. . . . . 715 Story Bldg., Los Angeles, Cal.  
 KREBS, A. SONNIN, 2nd Vice-Pres. and Treas., The Krebs Pigment &  
 Chemical Co., Newport, Del.  
 LAUGHLIN, SAMUEL W. . . . . Min. Engr., Consolidated Coppermines Co., Kimberly, Nev.  
 LAVERY, H. H., Supt., Thunder Mining Co., Ltd., St. Anthony Mine P. O., via  
 Fort William, Ont., Canada.  
 LLOYD, T. W., Asst. Engr., Land Dept., Tennessee Coal Iron & Railroad Co.,  
 1602 Brown Marx Bldg., Birmingham, Ala.

- MATSUZAWA, DENTARO, Supt. of Nishiyama Oil-field of Nippon Oil Co.,  
Nishiyama, Echigo, Japan.
- MORGAN, ROBERT J. . . . . Asst. Engr., Spassky Copper Mine, Ltd., Akmolinsk, Siberia.
- RIDDELL, J. MURRAY, Min. Engr., Chief Engr., Bates & Florence Iron  
Companies, Iron River, Mich.
- NICHOLS, NELSON NEWTON, Chief Engr., Delaware, Lackawanna & Western  
Railroad, Coal Min. Dept., 902 Monroe Ave., Scranton, Pa.
- OHNSORG, EDWARD L. . . . . Supt., Federal Lead Co., Federal, Ill.
- OLINGER, ROBERT WALLACE, Ore Buyer, W. R. Grace & Co., Rua Visconde  
de Inhauma 56, Rio de Janeiro, Brazil.
- PEYTON, W. C., Engr. . . . . 55 Liberty St., New York, N. Y.
- PLUMHOF, W. J. . . . . Engr. & Chemist, Caldo Mining Co., Frisco, Utah.
- POPE, ARTHUR C., Supt., Tiltonsville Plant, Prime Western Spelter Co., Care  
Prime Western Spelter X, Tiltonsville, Ohio.
- REYER, E. E., Supt., Matehuala Unit, American Smelters Securities Co.,  
Minas Dolores y Anexas, Matehuala, San Luis Potosi, Mexico.
- SAUVAJOL, PIERRE. . . . . Ingenieur conseil, 118 Boulevard Haussmann, Paris, France.
- SCHULTZ, R. S. JR. . . . . Cons. Engr., Marquette Cement Mfg. Co., La Salle, Ill.
- SMALL, WALTER M., Oil & Gas Geol., Vice-Pres. & Genl. Mgr.,  
Mid-Continental Development Co., 318 Ohio Bldg., Tulsa, Okla.
- SMITH, CLIFFORD ERNEST COVELL, Min. Engr., The Coniagas Mines, Ltd., and  
The Coniagas Reduction Co., Ltd., St. Catharines, Ont., Canada.
- SMITH, FRANK G. . . . . Physical Met., The American Brass Co., Waterbury, Conn.
- SUR, F. JOHN. . . . . 503 Commonwealth Bldg., Denver, Colo.
- SUZUKI, I. . . . . Min. Engr., Metal Mining Dept., Mitsubishi Co., Tokyo, Japan.
- VOLKER, HENRY JOSEPH, Min. Engr., Mine Foreman, Cerro de Pasco Min. Co.,  
Cerro de Pasco, Peru.

### *Junior Members*

- ALBI, CHARLES. . . . . 1650 Vine St., Denver, Colo.
- ANDREWS, CLARENCE W., Special Apprentice, American Steel Foundries,  
357 E. Market St., Alliance, Ohio.
- BLOGG, CECIL F. . . . . 708 Belmont Pl., Seattle, Wash.
- BUCKINGHAM, ALBERT. . . . . University of Nevada, Lincoln Hall, Reno, Nev.
- CASSILLY, THOMAS E., Min. Engr., E. J. Longyear Co., 710 Security Bldg.,  
Minneapolis, Minn.
- CHAN, ALBERT K., Student, Min. Engr., Colorado School of Mines, Golden, Colo.
- EISELE, LEWIS G. . . . . Min. Engr., City Engr., Box 164, Iron Mountain, Mich.
- ROBERTSON, FITCH, Student, Co. A. Engineers, Colorado National Guard,  
Pueblo, Colo.
- HARRINGTON, JOHN M., Min. Engr., Pickands, Mather & Co., Elba Mine,  
Box 516, Gilbert, Minn.
- HILL, A. B. . . . . Asst. to Oil Geol., 165 Harrison St., Clarksburg, W. Va.
- HOTCHKIN, HARRY. . . . . Min. Engr., 5522 Kenmore Ave., Chicago, Ill.
- KOROTKIN, WILLIAM, Student, Michigan College of Mines, 253 College Ave.,  
Houghton, Mich.
- LINK, MARCUS W., Student, University of Wisconsin, 215 North Park St.,  
Madison, Wis.
- MALLAMS, ARTHUR C., Min. & Civil Engr. . . . . Weir, Kans.
- MEAD, RICHARD, Student, Harvard College, 13 Wadsworth House, Cambridge, Mass.
- MURPHY, WILLIAM J. . . . . Box 441, Golden, Colo.
- NORTON, PAUL T. JR., Transitman, United Coal Corp.,  
First National Bank Bldg., Pittsburgh, Pa.
- ROBINSON, EDWARD W. . . . . Chemist, Aetna Explosives Co., Box 481, Oakdale, Pa.
- SMALL, SIDNEY S., Min. & Met. Engr. . . . . Ft. Monroe, Va.
- SMULLEN, C. KENNETH, 15th Company Engineers, 5th Provisional Regiment,  
Care Washington Barracks, Washington, D. C.
- THOMAS, RICHARD W. . . . . Student, Pennsylvania State College, State College, Pa.
- WILLIAMS, RALPH O., Special Apprentice, American Steel Foundries,  
357 E. Market St., Alliance, Ohio.

**Total Membership, July 10, 1917. . . . . 6,365**

## CANDIDATES FOR MEMBERSHIP

**APPLICATION FOR MEMBERSHIP.**—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

*Applications Lacking Endorsement*

Application for membership has been received from Mr. Mackay, whose record is given below. This application lacks the necessary number of endorsers, but since this candidate lives at some distance from the headquarters of the Institute, his record is published here in order that any members who are acquainted with him may be advised of the circumstances and may have an opportunity of writing to the Secretary endorsing this candidate.

**Alexander Dudley Mackay, Waratah, Tasmania.**

Proposed by J. D. Millen, Harold Rabling.

Born 1888, St. Lawrence, Queensland. 1903-06, Launceston Grammar School. 1907-09, University of Tasmania, B. Sc. 1910-12, University of Melbourne, B. M. E. 1916, Degree of M. M. E. 1912-13, Asst. to chief mining inspector, Mines Dept. of Victoria. 1913-16, Chief Assayer, engaged in research work, Mt. Bischoff Tin Mining Co.

Present position: Chief Assayer, Mt. Bischoff Tin Min. Co.

The following persons have been proposed during the period June 10, 1917, to July 10, 1917, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

*Members*

**William D. Armstrong, Empire, Ala.**

Proposed by Robert Hamilton, T. L. Nelson, H. S. Geisner.

Born 1884, Kosciusko, Miss. 1894-98, Public School, Dolomite, Ala. 1900-04, Grammar and High School, Dora, Ala. 1905, Business College, Birmingham, Ala. 1907-08, Special in Civil Engineering, Auburn, Ala. 1904, Rodman, mine surveying, Dora, Ala. 1906-07, Chainman and transitman, preliminary R. R. surveys, Tidewater Development Co., Birmingham, Ala. 1907, Rodman and inspector, R. R. construction, Louisville & Nashville R. R. Co., Birmingham, Ala. 1909-10, Surveyor, U. S. Engineers Office, Nashville, Tenn., topographic and hydrographic surveys. 1910-12, Asst. Constr. Engr., Tennessee Coal, Iron & R. R. Co., Birmingham, Ala., construction by-product coke ovens. 1912-13, Engr. highway constr., Noxubee Co., Miss. 1913-14, Surveyor, U. S. Engineers' Office, Chattanooga, Tenn. 1914-15, Jun. Engr., Interstate Commerce Commission, Chattanooga, Tenn. 1915-16, Draftsman, U. S. Engineers' Office, Florence, Ala. 1916-17, Asst. Erecting Engr., construction of by-product coke plant, Alabama City, Ala.

Present position: Resident Engr., Empire Coal Co.

**Irving Van Aken Augur, Los Angeles, Cal.**

Proposed by A. Eaton, R. B. Moran, C. M. Heron.

Born 1886, Le Mars, Iowa. 1912, Graduated, University of Cal. (Geol.) B. S. 1915-16, Lake View No. 2 Oil Co., Geol. 1916, Kern Trading and Oil, Geol. 1917, Geol., State Mining Bureau, California.

Present position: Oil Inspector, Cal. State Mining Bureau.

**Charles Lawrence Baker, Austin, Tex.**

Proposed by D. M. Phillips, L. L. Hutchison, E. L. Porch, Jr.

Born 1887, Cordova, Ill. 1903-05, Monmouth College, Monmouth, Ill. 1905, University of Minnesota. 1905-07, Oberlin College, Oberlin, Ohio, 1907-10, University of Chicago, Chicago, Ill. 1910-12, Univ. of California, Berkeley, Cal. 1908, S. B., University of Chicago. 1916, M. A., University of Cal. 1907, Mine Examiner, Oberlin Prospecting & Min. Co. 1908 and 1910, Field Asst., U. S. Geol. Survey. 1911, Instructor in Geol., University of Cal. 1912-13, Geol., Rio Bravo Oil Co. 1914, Geol. Nevada Petroleum Co. 1914, Geol., Bureau of Economic Geol., Univ. of Texas. 1915-16, Instructor in Geol., Univ. of Texas.

Present position: Geol., Bureau of Economic Geology, Univ. of Texas.

**Tom Lee Ball, Bessemer, Ala.**

Proposed by C. F. Schaber, H. S. Salmon, C. E. Abbott.

Born 1888, Florence, Wis. 1903-07, University High School, Birmingham, Ala. 6 mos. 1909-10-11, 2 mos. 1912, Washington and Lee University, Lexington, Va., Diploma in Geol. 1908-09, Transitman, Exploration Engr., Brown Ore Mines, T. C. I. & R. R. Co., Ore Mines. Mar. 1, 1912 to Aug. 20, 1912, Mine Sampler and Asst. Assayer, Live Oak Development Co., Miami, Ariz. Aug. 24, 1912, to Dec. 14, 1914, Operator Tables, Classifiers and Experimental Oil Flotation Machine, Miami Copper Co.

Present position—Feb. 1, 1915, to date: Efficiency Engr., Tennessee Coal, Iron & Railroad Co.

**Arthur E. Bendelari, Picher, Okla.**

Proposed by D. C. MacKallor, T. T. Read, W. H. Shearman.

Born 1879, Toronto, Ont., Can. 1888-95, Public Schools, Toronto, Ont. 1895-97, High School, Toronto, Ont. 1901, Asst. Mgr., Underwriters Land Co. 1901-03, 1903-11, Mine Operator having charge of Ground Floor M. Co., Mount-Nemo M. Co., and several others, Webb City, Mo. 1911-14, Mgr. of Mines for Underwriters Land Co.

Present position—1914 to date: Mgr. of Mines for Eagle Picher Lead Co.

**John Paty Benson, Tayoltita, San Dimas, Dgo., Mex.**

Proposed by Stuart L. Rawlings, R. H. Townsend, H. A. Buehler.

Born 1885, Honolulu, H. I. Public Schools, Cal. 1902, School of Mech. Arts, San Francisco. 1907, Univ. of Cal., B. S. 1908, Mill Supt., Assayer and Surveyor, San Vicente Min. Co., San Vicente, Mex. 1913, Mill Supt., San Luis Min. Co., Durango, Mex.

Present position: Mill Supt., San Luis Min. Co.

**Paul Cuno Berkefeld, Berkeley, Cal.**

Proposed by A. W. MacNichol, T. V. Reeves, P. S. Haury.

Born 1884, Santa Ana, C. A. 1899-1903, Cal. School of Mech. Arts. 1903-06, University of California. 1906-09, Instrumentman, Western Pacific Ry. Co. 1909-10, Supt., Four Hills Min. Co. 1910-13, Hydrographer and Asst. Engr., Duryea, Haehl and Gilman. 1913-14, Asst. Engr., Construction South San Joaquin Irrigation District. 1914-15, Independent work. 1915-16, Asst. Supt., Jackson Mt. Copper Co. 1916-17, Supt., Posey Hill Mining Co.

Present position: Mining Engr.

**C. J. Billwiller, Jr., Iquique, Chile.**

Proposed by C. R. Corning, W. L. Saunders, Huntington Adams.

Born 1884, Brooklyn, N. Y. 1901, Grad., St. Paul's School, Concord, N. H. 1906, Cornell Univ., M. E. 1906-08, Special apprentice and draftsman, Mechanical

Dept., Union Pacific R. R. shops, Omaha. 1909 to date, Engineering buyer and Engr. in Iquique, Chile, W. R. Grace & Co., N. Y. and allied companies.

Present position—1916 to date: Mgr. Oficina Paposo, Grace Nitrate Co.

**Matthew R. Blish**, Denver, Colo.

Proposed by H. Bancroft, D. W. Brunton, David Taylor.

Born 1889, Kewanee, Ill. 1907, Graduated from Kewanee, Ill., High School. 1911, Degree of B. M. E. from Univ. of Mich. 1912, Degree of M. S. E. from Univ. of Mich. June, 1912 to date, with Sullivan Machy. Co., as Asst. to Genl. Sales Mgr., and as Mgr., of Denver Office.

Present position: Mgr., Sullivan Machy. Co.

**Edward Howard Blum**, Philadelphia, Pa.

Proposed by R. P. Tinsley, P. W. Henry, H. W. DuBois.

Born 1886, Philadelphia, Pa. 1901–05, Philadelphia Central High School. 1905–09, Temple College, Philadelphia. 1905 to date, Engineering Dept. of The Atlantic Refining Co., Philadelphia, positions of rodman, transitman, on survey corps and draftsman, designing work and for five years in charge of the engineering dept. For last several months have acted as Asst. to the Pres. on engineering matters.

Present position: Engr. Asst. to the Pres., The Atlantic Refining Co.

**Gail Borden**, New York, N. Y.

Proposed by W. S. McKee, D. D. Moffat, William T. MacDonald.

Born 1872, Brewster, N. Y. 1889, Finished Public School, Wallkill, N. Y. 1889–92, Pratt Inst. Tech. High School, Brooklyn, N. Y. I am a mine owner not operating at present. My experience has been mostly in developing prospects. 1892–93, Engr. Corps, Baltimore Street Ry. Co., Baltimore, Md. 1893–95, Supt., Borden's Home Farm, Wallkill, N. Y. 1896–1900, Prospecting and developing mining properties in Cal. 1900–05, Pres., Frazier Borate Min. Co., Los Angeles, Cal. 1905–10, Prospecting and developing. 1910–14, Developing (my personally-owned) Min. Property in Trabuco Canyon, Cal. 1914–17, Inactive.

Present position: None.

**Walter M. Briggs**, Boston, Mass.

Proposed by R. L. Agassiz, T. T. Read, E. E. Bugbee.

Born 1871, St. Louis, Mo. 1895, Harvard University, A. B. Degree. 1890, Manual Training School, St. Louis, Mo. 1913–14, Sec.-Treas., Tennessee Copper Co.

Present position: Pres. Regal Mines Co.

**William H. Buell**, Wilmington, Del.

Proposed by G. C. Stone, F. S. MacGregor, L. D. Huntoon, F. N. Pease, L. W. Bahney, G. B. Street.

Born 1878, Philadelphia, Pa. 1899, Yale, Ph. B., Chem. 1900, P. R. R. Asst. M. M. 1905, Winchester Repeating Arms Co., Experimental Engr. 1917, Asst. Director, Experimental Station, Du Pont Co.

Present position: Asst. Director, Experimental Station, Du Pont Co.

**John Hayes Campbell**, Chicago, Ill.

Proposed by H. W. Nichols, L. V. Rice, R. W. Hunt.

Born 1871, Logosotee, Ind. 1888–91, St. Louis M. T. School, diploma. 1891–95, Washington Univ., St. Louis, Degree of B. S. 1895–97, Private chemical commercial laboratory and chemist, W. E. Fugo Co., Cincinnati. 1899–1900, Chemist, American Pipe & Foundry Co., Bessemer, Ala. 1900–01, Chemist and Engr., Roane Iron Co., Rockwood, Tenn. 1901–03, Asst. Supt., Sheffield Coal & Iron Co., Sheffield, Ala. 1903–05, Concrete Inspector and Timber Chemist, C. C. C. & St. Louis R. R., Cincinnati. 1905–06, Asst. Supt., Philip Carey Co., Cincinnati, Ohio. 1906–07, Private business asphalt roofing and flooring, Cincinnati, Ohio. 1897–99, Chemist, P. J. Ingle Co., Middletown, O. 1907–10, Supt. and Engr., Mine La Motte Lead & Smelting Co., Mine La Motte, Mo. 1910 to 1912, Robert W. Hunt & Co., Chemist and Chief Chemist, St. Louis. 1913, Chemical Engr., R. W. Hunt & Co., Chicago.

Present position: Chemical Engr., in charge of Laboratory R. W. Hunt & Co.

**William R. Canton**, McGill, Nev.

Proposed by Milnor Roberts, Joseph Daniels, C. E. Weaver.

Born 1888, Oniel, Nebr. 1907, Graduated from High School. 1911, Graduated from University of Washington with degree B. S. in Min. Engr. 1913, Asst. Engr.

Survey Municipality Cloverdale, B. C., under Mr. J. McHugh. 1914, Transitman, Constr. Pacific Highway, Jackson Co., Oregon (Medford) under M. O. Bennett. 1915-17, Millman, Alaska Gold Mill, Thane, Alaska under E. V. Daveler. 1917 (June), Nevada Consolidated Copper Co., as Millman, under Ross Taylor.

Present position: Rollman.

**Rowley J. Clarke**, Spokane, Wash.

Proposed by L. K. Armstrong, W. H. Linney, J. Mc D. Porter, F. A. Thomson.  
Born 1880, Rochester, N. Y. 1896-1900, Spent 4 years in the Spokane High, Graduated 1900. 1900-02, Two years in the mining school of the University of California. 1902, Six months as amalgamator, in 10 stamp, with 6 months surveying. 1903 to 1914, Levelman, transitman, asst. engr., railroad, Spokane International R. R. Co. 1905-08, General mine work, on patent surveys, assayer in employ of my brother Roy H. Clarke. 1909, Special litigation underground surveyor for the Bunker Hill & Sullivan Mining Co. 1910, Park Engineer, City of Spokane, Wash.

Present position—1918 to date: General Mgr., Iron Creek Min. Co.

**Bernard Langhorne Cope**, Goldfield, Nev.

Proposed by J. W. Hutchinson, H. M. Rives, J. B. Lain.  
Born 1889, Wiltshire Co., Eng. 1907, Grammar schools of Orange Co., Cal. and graduate of Santa Ana High School, Orange Co., Cal. 1914, Graduate of College of Natural Sciences, University of California. 1911-13, Operator and foreman, Nevada Hills M. & M. Co.'s mill, Fairview, Nev. 1914-15, Cyaniding tailing in Colombia, So. America. 1916-17, Operator and Foreman, Goldfield Cons. Mill, Goldfield, Nev.

Present position: Foreman, Goldfield Cons. Mill.

**Fred S. Curtis**, Cleveland, Ohio.

Proposed by F. R. Van Horn, A. W. Smith, Z. Jeffries.  
Born 1878, White Pigeon, Mich. 1896, Graduated from High School. 1898-1900, Michigan Agric. College, Mechanical Engineering Course. 1905-06, University of Utah. Special Mining Engineering Course. No Degree. 1900-05, Mining occupation, including construction work (Inspector), Michigan, Lake Superior Co., surveying, mining, railroad surveying and construction, steam plant operation, etc. 1907-11, Steptoe Valley Mining & Smelting Co., Engineering Dept., during construction of smelter and concentrator, chemical laboratory, and 2 years in charge sampling and research of concentrator. 1911, 4 months with Miami Copper Co. as concentrator "shift boss." 1911-17, 2 years traveling in Mex. and Western U. S. for the W. S. Taylor Co., afterward in charge of research work for same company.

Present position: General research in uses of wire cloth for same company.

**William L. Curtis**, Bradford, Pa.

Proposed by P. N. Moore, W. L. Bell, Victor Rakowsky.  
Born 1848, Corning, N. Y. I have been actively interested in the production of oil for many years, and at the present time am counted as an oil producer in the States of New York, Illinois, Kansas, Oklahoma, Louisiana, Pennsylvania, Ohio, Indiana, California and in Mexico, always having operated in my own interest with other business associates. In gold mining I have had experience as a stockholder in the Inca Min. Co. of Peru, The Oro Fino Mining Co., of which I am now President, located in Placer Co., Cal., and as stockholder in the Baltic Mining Co., Webb City, Mo., The Queen Esther Min. Co., Webb City, Mo., and the Admiralty Zinc Co., Quapaw, Okla.

Present position: Pres., The Oro Fino Mining Co., Placer County, Cal.

**R. H. Eagles**, Palmerton, Pa.

Proposed by L. S. Holstein, T. T. Read, W. M. Kelsey.  
Born 1895, Madison, N. J. 1915, A. B., Cornell University. 1915-16, Phelps-Dodge Copper Co. (Copper Queen Branch). 1916, New Jersey Zinc Co.

Present position: Chemist, New Jersey Zinc Co.

**Edgar Wallace Engle**, North Chicago, Ill.

Proposed by H. H. Stock, E. A. Holbrook, C. M. Young.  
Born 1888, Buffalo, Mo. 1912, B. S., Drury College. 1914, M. S., University of Illinois. 1916, Ph. D., University of Illinois, in Chemistry. 1912-15, Asst. in chemistry, University of Illinois. 1915-16, Fellow in Chemistry, University of Illinois. 1916-17, Instructor in Chemistry, University of Illinois.

Present position: Research Chemist, The Pfanstiehl Co.

**Marion Henry Foss, Salt Lake City, Utah.**

Proposed by V. S. Rood, C. W. Whitley, R. C. Gemmell.

Born 1885, Wilmette, Ill. 1904-08, Massachusetts Institute of Tech. 1908-09, Leland Stanford Junior Univ. 1909-10, Mackay School of Mines of the Univ. of Nev., B. S. 1911, Underground at Goldfield Cons. Mines Co., Goldfield, Nev. In mill at National Mines Co., National, Nev. 1912, In charge of Cobalt Smiley Mining Co., North Cobalt, Can. In mill at Maricopa Mines Co., Austin, Nev. Engr. for Gold Note Mining Co., Kennedy, Nev. 1913-16, Engr. for Utah Apex Mining Co., Bingham Canyon, Utah. 1916-17, Engr. Utah Metal & Tunnel Co.

Present position: Min. Engr.

**Sidney Longman Galpin, Ames, Iowa.**

Proposed by S. W. Beyer, L. E. Young, H. Ries.

Born 1886, Jefferson, Ohio. 1903, Graduated from High School, Jefferson, Ohio. 1907, A. B., Adelbert College (of Western Reserve U.). 1907-12, Graduate student at Cornell Univ., economic geol., petrography and mineralogy. 1910, A. M., Cornell Univ. 1912, Ph. D., Cornell Univ. 1907, Asst. in geol., Cornell Univ. 1911, Instructor in geol. Cornell Univ. 1910, Junior Geol., U. S. G. S. (summer only). 1911 (summer), Instructor in charge of Oberlin geological field camp, Virginia. 1912-14, Asst. State Geol., Georgia. 1914, Asst. Prof. geol. and min., Iowa State College. 1916 (summer), Petroleum Geol., Pierce Oil Corp'n.

Present position: Asst. Prof. Geol. and Min., Iowa State College.

**Albert E. Gregory, Georgetown, Cal.**

Proposed by C. W. Goodale, W. J. Sharwood, C. W. Merrill.

Born 1879, England. To 1896, St. John's College, Hurstpierpoint, Sussex, England. 1906-08, Colorado School of Mines, Golden, Colo. (Course not completed.) 1897-1901, Office, Exploration Co., Ltd., London, Eng. 1901-06, Accountant, Montana Mining Co., Ltd., Marysville, Mont. 1908-12, Assayer, sampler and examinations, Tomboy Gold Mines Co., Ltd., Telluride, Colo. 1912-16, Assayer, mine engr. and examinations, Smuggler, Union Mining Co., Telluride, Colo. 1916 to date, Georgetown, Cal.

Present position: Supt., Beebe Mine, with Bulkeley Wells, Agent.

**William Evans Guy, St. Louis, Mo.**

Proposed by P. N. Moore, H. A. Wheeler, G. O. Carpenter.

Born 1844, Cincinnati, Ohio. 1865, Graduated Princeton, A. B. 1869, Degree A. M., Princeton Univ. 1865-66, Heidelberg, Chemistry. 1866-69, Freiberg, Germany, Student General Min. Engrg. and Met. 1869-70, General Min. Engrg. and visiting western mines. 1870-71, Asst. Supt., Stuart Reduction Works, Georgetown, Colo. 1871-72, Asst. Geol., Mo. Geol. Survey. 1872-77, Vice-Pres., St. Louis Bolt & Iron Co., St. Louis, Mo. 1877-79, Studied Law, Cincinnati Law School, LLB. 1879-85, Pres. and Vice-Pres., Tudor Iron Works, St. Louis. 1886-89, Traveled in Europe, U. S. and Canada. 1889-1903, Organized and built St. Louis, Peoria & Nor. Ry., St. Louis Valley Ry., St. Louis & Gulf Ry. 1889-1903, Built and operated six Coal Mines in Ill. (Madison Coal Co.). 1911, Organized Laclede Steel Co., St. Louis, Mo. 1917, LL.D., Westminster College, Fulton, Mo.

Present position: Retired.

**Clyde Willis Hall, Metcalf, Ariz.**

Proposed by W. S. Hall, W. H. Seamon, Jr., C. R. Forbes, H. H. Nowlan.

Born 1889, Pleasanton, Kans. 1906-09, Pleasanton High School, Pleasanton, Kans. 1909-11, Kansas State Agricultural College, Manhattan, Kansas, one year prep. and one year engineering. 1911-14, Missouri School of Mines, Rolla, Mo. Graduated with degree of B. S. in Min. Engrg. 1913, Northern Gold Reef, Ltd., Mining Co., St. Anthony, Ont., Canada. With company three months as engr. 1914-16, Yoquivo Development Co., Yoquivo, Chih., Mex., Engr., Assayer and Supt. 1916, Calumet & Sonora Mining Co., Box 35, Cananea, Son., Mex. With company six months as engr. and chemist.

Present position—1916 to date: Mine Shift-Boss, Arizona Copper Co.

**W. H. W. Hamilton, Sumpter, Oregon.**

Proposed by L. K. Armstrong, W. H. Linney, J. C. Haas.

Born 1863, England. High School Education. 1879, Apprenticed to Mining Engr., at Dunedin, New Zealand. 1886, Finished apprenticeship. Since then have always been in charge of mines. Am now manager of the January Min. Co., and the



Hamilton Mines Co., and largest owner in both properties. Have always reserved the right to examine and report on mining properties, and have had lots of it to do. 1888, Went to Arizona and contracted mining work for several years, part of the time working as assayer or surveyor. 1896, Supt. for 12 months for White Hills M. M. Co., at White Hills, Ariz., then next 12 months manager, for same company. Resigned to accept management of Sheep Trail Mines, 30 miles from White Hills. 1899-1904, Mgr. at Sumpter Mines.

Present position: General Mgr., Hamilton Mining Co.

**Chester Armstrong Hammill**, Tulsa, Okla.

Proposed by R. A. Conkling, L. L. Hutchison, F. J. Fohs.

Born 1889, Chicago, Ill. Sept., 1908, to Dec., 1909, University of Illinois, Dept. of Mech. Engrg., Jan., 1910, to June, 1915, University of Chicago; S. B. June 10, 1913. Undergraduate work in Chemistry and Geol. Graduate work in Geol. and Palaeontology. Aug. 1912, Field course, Baraboo Region, Wisconsin. Sept. 1912, Field course, Cimarron Quadrangle, Colo. Sept. 1914, Field course, Ste. Genevieve Co., Mo. June to Aug., 1915, Geol., Wisconsin State Geological Survey, on survey of Penokee Iron Range. Sept. 1915, Geol., Roxana Petroleum Co. of Okla. Petroleum examinations in Kansas, Oklahoma and Texas.

Present position: Geol., Roxana Petroleum Co.

**Robert W. Handley**, Anaconda, Mont.

Proposed by S. S. Rodgers, A. E. Wiggin, B. S. Morrow.

Born 1890, Alamosa, Colo. 1904-07, High School, Carnegie, Pa. 1908-12, School of Mines and Metallurgy, Pa. State College, B. S. 1912-15, Resident Min. Engr., The Shenango Furnace Co., Chisholm, Minn. 1915-17, Asst. Testing Engr. and Foreman of Zinc Concentrator, Anaconda Copper Min. Co., Washoe Reduction Works, Anaconda, Mont.

Present position: Foreman, Zinc Concentrator, Anaconda Copper Mining Co.

**Alfred Tirrell Hastings**, Bingham Canyon, Utah.

Proposed by Geo. A. Packard, Arthur D. Knowlton, Frederick Lyon.

Born 1889, So. Weymouth, Mass. 1900-06, Milton Academy, Milton, Mass. 1906-08, Roxbury Latin School, Roxbury, Mass. 1910-12, Machine man underground, Utah-Apex Min. Co., Bingham, Utah. 1912, Short time underground, Nevada Douglas Copper Co. 1913, 6 months, Kirk & Leavell, Salt Lake City, Utah. 1914-15, 6 months office man, 9 months in mines, Aurora Cons. Mines Co., Aurora, Nev. 1915-16, Big Four Exploration Co., Park City, Utah. 1916-17, Bingham Coalition Mines Co., Bingham, Utah.

Present position: Mgr., Bingham Coalition Mines Co.

**Jerome A. Hilbert**, Cornucopia, Ore.

Proposed by Frank S. Baillie, Willard H. Nash, G. Edward Kedzie.

Born 1882, Creve Coeur, Mo. 1891-96, Public Grammar School. 1906, I. C. S., Metal Min. Course. 1904-09, Sampler, Shift boss, Mine Foreman, Asst. Supt., Cia. Metalurgica-Mexicana, Sierra Mojada, Mex. 1909-10, Foreman, R. S. Towne's Interests, San Luis Potosi, Mex. 1910-13, Local Supt., Cia. Metalurgica-Mexicana, Sierra Mojada, Mex. 1913-14, Foreman, Tezuatlan Copper Co., Tezuatlan, Mex. 1914-16, Miner, Joplin, Mo.; Butte, Mont. Sampler, Valmy, Nev.; Black Butte, Ore. Foreman, Snake River Mines, Ore.

Present position: Mine Foreman, Union Companion Mine.

**Elias Hogberg**, North Kamloops, B. C., Canada.

Proposed by Frederic Keffer, Rowland King, Arthur P. Allen.

Born 1883, St. Petersburg, Russia. 1892-98, General education equal to High School in Canada, acquired in Sweden. 1898-1900, Two years in manufacturing plant of "Husgvarna Vapenfabrik," Sweden. 1901-06, General construction work in wood, steel and concrete in Canada with Barnett & McQueen Co., John A. Mead Constr. Co., and other construction companies, mainly in Manitoba and Ontario. Port Arthur and Fort William, millwrighting in saw mills, two seasons. 1907-13, Contracting in bldg. line, and manufacturer of canvas goods, at Port Arthur. 1914, Eight months as millman at Iron Mask Mine (Kamloops Copper Co.,) B. C. 1915-16, Prospecting in B. C. 1916-17, Highland Valley Mining & Development Co., B. C.

Present position: Mill Foreman.

**Elgin Bryce Holt, Phoenix, Ariz.**

Proposed by Fred H. Perkins, A. F. Muter, F. W. Libbey.

Born 1873, Harrison, Ark. 1897, Grad. State College, New Mex., B. S., 1897-1902, Prospecting. 1902, Post Graduate Course, State College, New Mex. 1903, Conducted Assay Office, Douglas, Ariz. 1904, Prospecting. 1905-11, Member firm of Holt Bros., Min. Engrs., Magdalena, Sonora, Mex. 1912-15, Pres. and Gen'l Mgr., Cerro de Plata Mines Co., S. A., Sonora, Mex. 1915, Operated in Yuma Co., Ariz. 1916, Operated in Santa Cruz Co., Ariz. 1917, Established in Phoenix, Ariz. Gen'l Cons. business in Min. Engineering.

Present position: Gen'l. Cons. Business.

**Lapsley William Hope, Ymir, British Columbia.**

Proposed by J. Gordon Hardy, J. M. Brooks, Jr., Ewing Carter, W. T. James.

Born 1881, Chattanooga, Tenn. 1896, Grad., Grammar School; 1899, Grad., High School, Chattanooga, Tenn. 1903, completed four years in special engineering course, Univ. of Tenn. 1904-08, Charge meter and testing dept., Chattanooga Electric Co. 1908-12, Elect. engr., power plant, Chattanooga Ry. & Light Co. 1912-13, Resident mgr. and supt., Credo-Eureka Min. Co., Eureka, Nev. 1913-14, resident mgr. and supt., Adams Hill Corp'n., (succeeded above). 1914 (Fall), Hobson Silver-Lead Co., Ltd., hydro-electric constr., Ymir, B. C. 1915-16, Member of firm and mgr., Buchanan & Gilder, genl. contractors, El Paso.

Present position—1916 to date: Supt. and assayer, Hobson Silver-Lead Co., Ltd.

**William Henry Janney, Hurley, New Mex.**

Proposed by D. C. Jackling, L. S. Cates, John M. Sully.

Born 1865, Salt Lake City, Utah. Common School, 1879-80, Nicholas Trewick, Park City, Utah. 1880-83, Lowell Min. Co., Park City, Utah. 1883-85, U. S. Marshall, E. A. Ireland, Salt Lake City, Utah. 1885, R. G. & W. Railway. 1886, Bingham Canyon. 1886-87, Ontario Min. Co., Park City, Utah. 1887, U. S. Marshall, Frank Dyer. 1887-89, Lead Mines & Mill. Co., Bingham, Utah. 1889, D. & R. G. Ry., Salt Lake City, Utah. 1889-94, Salt Lake Rapid Transit Co. 1894, Electrical Firm, Janney & Hurd, Salt Lake City, Utah. 1894-96, Eureka Electric Light Plant, Eureka, Utah. 1896-98, Eureka Hill Min. & Mill. Co., Eureka, Utah. 1898, Capt. De Lamar, D. C. Jackling, Duncan MacVichie, John Durn, F. G. Janney 1901, U. S. Mine & Smelter. 1902, Bamberger-De Lamar, Simon Bamberger, at De Lamar and Kershaw, Nev. 1899-1900, Gold Mountain, Cal. 1903-11, Utah Copper Co., Bingham, Copperton, Magna and Arthur, Garfield. 1911-12, Ray Consolidated Copper Co., Hayden, Ariz. 1912-17, Chino Copper Co., Hurley, New Mex.

Present position: Supt. of Mills.

**Olaf Pitt Jenkins, Pullman, Wash.**

Proposed by L. K. Armstrong, S. Shedd, J. C. Branner.

Born 1889, Greencastle, Ind. 1913, Stanford Univ., A. B.; 1915, M. A. 1911, Asst. Geol. Stanford Expedition to Brazil. 1913-16, Asst. Geol. Tenn. Geol. Survey. 1916, Asst. Geol., Gypsy Oil Co., Tulsa, Okla. 1916-17, Asst. Prof. of Econ. Geol., State College of Wash., Pullman, Wash. 1917, Asst. Geol., Washington Geol. Survey.

Present position: Asst. Prof. of Economic Geol. State College of Wash.

**John A. Jess, Carthage, Mo.**

Proposed by Howard I. Young, Charles T. Orr, A. W. Koch.

Born 1879, Springfield, Ill. 1903, Princeton Univ., C. E. 1904-05, Colorado State School of Mines. 1906, General underground. 1907-08, Supt., Rincon Min. Co. 1908, with Harry S. Lee. 1909, Supt., Nevada, Ill., N. & M. Co. 1910-14, Field Engr. and Mine Supt., H. L. Hollis and Elco Min. Co. 1915, General practice. 1915-17, Supt. of mines, American Zinc, Lead & Smelt. Co.

Present position: Gen'l Supt., Commonwealth Lead &amp; Zinc Co., Picher, Okla.

**Mead Samuel Johnson, Miami, Okla.**

Proposed by D. D. Dunkin, J. C. Reid, G. M. Brown.

Born 1888, Wichita, Kansas. 1907, Oklahoma A. & M. College. 1908, Okmulgee High School. 1909, Missouri School of Mines. 1910-12, Oklahoma School of Mines. State Board of Education. Oklahoma City, Okla. Indianoma Refining Co., St. Louis, Mo. Okmulgee Cons. Oil Co., Okmulgee, Okla. 1912, Professor, Mining Dept., State of Okla. 1913, Professor Mining Extension, State of Okla. 1913, Professor, Okmulgee Consolidated Oil Co. 1914-16, Professor, State of Okla., Mining Extension. 1916-17, Geol., Indianoma Refining Co.

Present position: Professor Mining Extension, State of Okla.

**Leonard Larson**, McGill, Nev.

Proposed by C. B. Lakenan, R. E. H. Pomeroy, J. C. Kinnear.

Born 1887, Salt Lake City, Utah. 1904-07, High School, Salt Lake City, Utah. 1907-11, Univ. of Utah, B. S. in General Engrg. with major work in Mining and Met. 1905, Coal Mining, Kemmerer Coal Co. 1906, Railroad Surveying, Oregon Short Line R. R. 1909, Coal Mining, Diamondville Coal Co. 1910, Metal Mining, Daily West Mining Co. 1911-16, Experimental work and assaying, Nevada Consolidated Copper Co. 1916-17, Roaster Foreman, Nevada Consolidated Copper Co.

Present position: Roaster Foreman.

**Jeremias Hendrikus Ledebver**, Newcastle, N. S. W.

Proposed by G. D. Delprat, David Baker, R. A. Field.

Born 1883, Borne, (O) Holland. 1895-1900, High School. 1900-05, Grad., Govt. Univ. of Engrg., Delft, Holland. 1905-06, Designing steam engines, K. & Th. Moeller, Brackwede, Westfalia. 1906-08, Charge of drawing office and workshop, Machinefabrick Maranatha, Rotterdam. 1908-11, Designing steam turbines, condensing plant, centrifugal pumps, Gebr. Stork & Co. 1911-15, Engr., Broken Hill Prop. Co., Ltd., Broken Hill, N. S. W.

Present position: Mech. and Chief Engr. of Construction, Steel Works, Broken Hill Prop. Co., Ltd.

**James Jerome Lillie**, Frisco, Utah.

Proposed by W. H. Hendrickson, E. R. Lidvall, Ernest Gayford.

Born 1893, Omaha, Neb. 1909-11, High School, Springfield, Mo.; 1912, Salt Lake City. 1912-16, Univ. of Utah, B. S. 1914, Asst. Engr., Consolidated Copper Mines Co., Kimberley, Nev. 1916, Asst. Testing Engr., International Smelting Co., Tooele, Utah. 1916-17, Mine Engr., Horn Silver Mines Co., Frisco, Utah.

Present position: Shift Boss, Horn Silver Mines Co.

**Curtis Lindley, Jr.**, East Ely, Nev.

Proposed by G. C. Riser, Jr., C. B. Lakenan, R. E. H. Pomeroy.

Born 1883, Stockton, Cal. 1902-03, Univ. of Cal. 1903, Melones Mine, Stanislaus, Cal. 1904-05, Bunker Hill & Sullivan, Ida. 1905-06, Nevada Cons. Mines Co., Ely, Nev. 1906, Tacoma Smelting Co. 1907, Northern Pacific R. R. Co. 1907-08, Cumberland Ely Copper Co. 1909, Giroux Cons. Copper Co.; Inspiration Copper Co. 1910, Original Amador Gold Min. Co. 1911-12, Big Flat Hydraulic Min. Co. 1913-15, Panama Pacific Exposition.

Present position: Engr. in charge of design and construction, crushing plant, Nevada Cons. Copper Co.

**Preston Locke**, Spokane, Wash.

Proposed by L. S. Cates, H. A. Guess, L. K. Armstrong, T. T. Read.

Born 1888, Pasadena, Cal. 1903, Classical School for Boys, Pasadena, Cal. 1903-04, Berkeley School, New York City. 1904-05, Kelvin School, New York City. 1905-09, Columbia School of Mines, New York City, Degree of E. M. Illig Medal in Mining. 1910-11, Surveyor and Engr., No. 1 Mine, Ray Consolidated Copper Co. 1911, Sampler and Timberman, Ray Cons. Copper Co. 1912, Stoping Engr. and Statistician, No. 2 Mine, Ray Cons. Copper Co. 1912-13, Geol., Ray Cons. Copper Co. 1913-14, Geol., Braden Copper Co. 1914, Shift boss, Braden Copper Co. 1915, Mine Foreman, Braden Copper Co. 1915-16, Field Engr., for Sherwood Aldrich, South American examinations.

Present position: Resident Engr. and Mgr., Northwestern Dept., Mining Dept. of the A. S. & R. Co.

**Fred James Longworth**, Greenwood, B. C.

Proposed by Oscar Lachmund, L. K. Armstrong, C. W. Goodale.

Born 1880. 1898-1901, Ortonville, Minn., H. S. 1901-05, Univ. of Minn. B. S. 1906-07, Asst. Chemist, B. C. Copper Co. 1907-08, Instructor, Univ. of Iowa. 1908-10, Chief Chem. and Met., B. C. Copper Co. 1910-13, Supt., Napoleon Mine, B. C. Copper Co. 1914, In business for myself.

Present position—1915 to date: Supt. Smelter, B. C. Copper Co.

**Walter W. Lytzen**, Butte, Mont.

Proposed by B. H. Dunshee, C. L. Berrien, Earl B. Young.

Born 1882, St. Paul, Minn. 1898-1901, St. Paul Central High School. 1901-05, School of Mines, Univ. of Minnesota. To 1905, Summers, Railway Surveying,

Northern Pacific and Great Northern Rys. 1905-06, Underground work, Butte, Mont., Asst. Chemist, Pittsmon Smelter, Butte, Mont. 1906-07, Mechanical Dept., Washoe Smelter, A. C. M. Co., Anaconda. 1907-09, Asst. Min. Engr., Butte & Boston Cons. Min. Co., Butte, Mont.

Present position—1910 to date: Engr., Anaconda Copper Mining Co.

**Tom Maslin**, Eureka, Utah.

Proposed by Walter Fitch, Cecil Fitch, C. W. Stimpson.

Born 1871, Redruth, England. Scranton School of Correspondence, 1901. General Mining Works, Great Cobar, Ltd., Australia. Yampa Smelting Co. B. & M. Smelting Co., Great Falls, Mont.

Present position: Master Mechanic, Chief Con. Mining.

**George Milliron**, West Winfield, Pa.

Proposed by Robert Peele, E. J. Hall, W. Campbell.

Born 1869, Bradys Bend, Pa. Grammar school education. Pursued a home course of study in mining and mechanical engineering. 1888-1900, Supt. of limestone mines and quarries for A. G. Morris & Son.

Present position—1900 to date: Genl. Supt. of limestone mines for Pittsburg Limestone Co.

**Melville Fuller Peters**, De Pue, Ill.

Proposed by J. E. Thomas, D. S. Grenfell, G. S. Brooks.

Born 1892, Ishpeming, Mich. 1911, Ishpeming Public Schools. 1911-13, Michigan Agricultural College. 1913-17, Michigan College of Mines. 1911-13, Marquette Co. Gas & Electric Co.

Present position: Research Dept., Mineral Point Zinc Co.

**Orrin P. Peterson**, Minneapolis, Minn.

Proposed by Edwin C. Holden, Julius Segall, Pope Yeatman.

Born 1886, Moscow, Wis. Common Schools in Wis. 1906-12, Univ. of Wis. 1914, Min. and geol. courses leading to professional degree. 1910, summer, Field work, Cleveland-Cliffs Iron Co., Ishpeming, Mich. 1911, Charge of field party, Oliver Iron Min. Co., Lake Superior Dist. 1912-14, Asst. Geol., Oliver Iron Min. Co. 1914-16, Geol., Adbar Dev. Co., Deerwood, Minn.

Present position—1916 to date: Private work for various companies in Canada, United States, Cuba.

**J. Edgar Pew**, Tulsa, Okla.

Proposed by A. F. Lucas, I. N. Knapp, W. E. Saunders.

Born 1870, Mercer, Pa. High School and Business College. No degrees. 1886-96, Peoples Natural Gas Co., Pittsburgh, Pa. 1896-1912, Sun Oil Co., and Sun Co., Philadelphia, Pa. and Toledo, Ohio. Located at Toledo, Ohio, and Beaumont, Tex. 1912-14, For self in Okla. 1914 and continuously to date, Carter Oil Co., Tulsa, Okla.

Present position: Vice-Pres. and Mgr., Carter Oil Co.

**George W. Potter**, Picher, Okla.

Proposed by D. C. MacKallor, T. T. Read, W. H. Shearman.

Born 1892, Galena, Kansas. 1908, Public Schools, Joplin, Mo. 1910-11, Special work, University of Missouri. 1909, Rodman and Levelman, Mitchell-Cooke Engr. Co. 1910, Storm and sanitary sewer work, W. K. Palmer Engr. Co. 1912-14, Water Supt., La Feria Mutual Canal Co., La Feria, Texas. 1915, Mining Supt., Eagle Picher Lead Co.

Present position: Genl. Supt. of Mines, Eagle Picher Lead Co.

**Roy S. Rhoades**, Casper, Wyo.

Proposed by Arthur Eaton, T. S. Harrison, L. D. Godshall.

Born 1891, Missoula, Mont. 1915, A. B. degree, Univ. of Cal., Geology. At present, Geologist for Western Exploration Co.

Present position: Oil Geologist.

**Charles William Shannon**, Norman, Okla.

Proposed by F. Julius Fohs, L. L. Hutchison, V. H. Hughes.

Born 1879, Atlanta, Ind. 1906, Ind. Univ., A. B. 1907, A. M. 1907-08, Post. Graduate Work. 1905, Indiana Geol. Survey. 1906, Geol. for C. I. & E. R. R. 1907, Asst., Ind. Geol. Survey. 1908, Asst., Ind. Geol. Survey. Sept., 1908-June,

1911, Head, Science Dept., Brazil, Ind., High School. 1908-11, Summers, Asst., Ind. Geol. Survey. 1911-13, Field Geol., Okla. Geol. Survey. 1911, Fall Semester, taught Geol., Okla. Univ.

Present position: Director, Okla. Geol. Survey.

**Stanley Adnan Spellmeyer**, Tayoltita, San Dimas, Dgo., Mex.

Proposed by Stuart L. Rawlings, R. H. Townsend, H. A. Buehler.

Born 1886, Kansas City, Mo. 1913, Grad., Univ. of California, B. S. 1907, Engrg. Dept., Atchison, Topeka & Santa Fe Ry. 1909, Assayer, Calumet & Arizona Min. Co. 1910, Engr. and Assayer, Tomita Min. Co., S. A., of San Felipe, Son., Mex. 1911, Surveyor, Kern Trading & Oil Co., Maricopa, Cal. 1913-15, Miner, millman and engr., Nevada Hills Min. Co., Fairview, Nev.

Present position—1915 to date: Min. Engr., San Luis Min. Co.

**Albert P. Spooner**, Bethlehem, Pa.

Proposed by W. L. Cumings, W. R. Shimer, F. O. Kichline.

Born 1887, Harrisburg, Pa. Graduate of Lehigh University, receiving degree of Met. Engr., Class of 1911. 1911, Penna. Steel Co., Steelton, Pa., Blast Furnace Dept. 1912-13, Asst. Supt., Supt. of Washeries, Canadian Collieries Co., Union Bay, Can. 1911, Surveyor, Kern Trading & Oil Co., Maricopa, Cal. 1913-15, Miner, millman and engr., Nevada Hills Min. Co., Fairview, Nev.

Present position—1916 to date: Asst. in Met. Dept., Bethlehem Steel Co.

**Charles D. Stark, Jr.**, Johnsville, Plumas Co., Cal.

Proposed by F. D. Pagliuchi, Philip Wiseman, M. P. Dalton.

Born 1881, Scotland. 1896, Common Schools in Scotland and U. S. 1896-1900, Technical School. 1900-04, Studied surveying, assaying, ore dressing and other subjects relative to mining. 1905-09, Asst. Supt. and Supt., Johnston-Graham Mining Co. 1909-10, Millman and mill foreman, Plumas-Eureka Mines Co. 1910-13, Asst. Supt., Plumas-Eureka Mines Co. 1913-15, Resident Mgr., Plumas-Eureka Mines Co.

Present position—1916 to date: Mgr., Plumas-Eureka Mines Corpn.

**Lloyd Lincoln Stewart**, Los Angeles, Cal.

Proposed by H. V. Welch, W. A. Schmidt, H. W. Morse.

Born 1889, Peru, Indiana. June, 1914, B. A., Stanford University, Dept. Chem. May, 1916, M. A., University of California, Dept. Geology. Jan.-May, 1917, Graduate Student, University of California, Dept. of Mining. 1914-15, Teacher of Mathematics, Santa Barbara, Cal. 1915-16, Student University of Cal. Summer, 1916, Chemist, General Chemical Co., Bay Point, Cal. August, 1916, to Jan., 1917, Private Laboratory, Los Angeles, Cal. Jan. 1917, to May, 1917, Graduate Student, University of California.

Present position: Chemist, Western Precipitation Co.

**Verne Ancil Stout**, San Francisco, Cal.

Proposed by P. S. Haurry, E. Jussen, R. E. Cranston.

Born 1887, Tehama, Cal. 1905-09, Univ. of Cal., B. S. 1909-12, Asst. Engr. Pato Mines (Columbia) Ltd. Zaragoza, Columbia, S. A. 1912, Practical dredging. Six months Natomas Co. of Cal., Natoma, Cal. 1912-13, Field Engr., Hammon Engrg. Co., Marysville, Cal. 1913, Examination in Mex. for W. P. Hammon, San Francisco, Cal. 1914, Met., Star Peak M. Co., Humboldt, Nev. 1914-17, Western Representative in San Francisco, Cal., for Hardinge Conical Mill Co.

Present position: Western Mgr. in the San Francisco office of Hardinge Conical Mill Co.

**Bruce White**, Sandon, B. C., Canada.

Proposed by L. K. Armstrong, J. McD. Porter, F. A. Thomson.

Born 1863, Ontonagon, Mich. 1870-1883, Common Schools of Michigan. 1884-1917, Mining in Michigan, Wisconsin, Washington, Queen Charlotte Islands, Mexico, Southern British Columbia, as miner, foreman, superintendent, and mine manager. For 15 years supt., Slocan Star Mine, B. C. For 6 years manager Noonday Mine, Slocan Dist., B. C. 1916-17, Chairman Western Branch, Canadian Mining Institute.

Present position: Mine Mgr.

**Sydney L. Wilkins**, Oronogo, Mo.

Proposed by Howard I. Young, Otto Ruhl, Charles T. Orr.

Born 1879, Gloucester, England. General High School. 1899-1901, Lucky Fire

Min. Co. 1901-04, with Frank Nicholson. 1904-08, Little Princess Min. Co. 1909-15, Deering L. & L. Co.; Dirigo L. & L. Co. 1915-17, Oronogo Circle Min. Co.  
Present position: Genl. Supt., Oronogo Circle Min. Co.

**Homer Oliver Williams, Pittsburgh, Pa.**

Proposed by R. B. Woodworth, H. P. Tiemann, W. MacGilvray Shiras.

Born 1888, Woodville, Pa. 1905, Pittsburgh High School. 1907, Pennsylvania State College. 1913, Carnegie Institute of Technology. At Pa. State College I took a short course in Mechanical Engrg. While at Carnegie Tech. I had a partial course in Met. and Chem., completing same by tutoring and subsequent work in steel mills. 1907, Started to work for Carnegie Steel Co., and have been associated with them ever since. During this period, I have filled the majority of positions in Rail and Billet Sales Dept. Have spent over three years in mills at Homestead open-hearth dept. and completed Carnegie Steel Co. Post Graduate Course which enabled me to work in every dept. of the Duquesne Steel Works. This was followed by my being appointed Asst. Instructor in Educational Bureau, which put me in touch with all phases of iron and steel manufacture.

Present position: Clerk in Specialty Dept.

**Robert H. Wood, Washington, D. C.**

Proposed by E. De Golyer, M. L. Lee, M. L. Thomas.

Born 1887, Wallaceburg, Ark. 1907-11, University of Okla., A. B. Degree. 1911-17, Junior and Associate Geologist, U. S. Geol. Survey.

Present position—1917: Consulting Petroleum Geologist.

### *Associate Members*

**T. Bircharde Kenvin, New York, N. Y.**

Proposed by R. Peele, R. M. Raymond, W. Campbell.

Born 1894, McAdoo, Pa. 1911-13, University of Wis. 1913-15, Columbia Univ. 1916-17, Columbia University, E. M. degree. 1915-16, Surveyor and Asst. Mgr. of Gorumahisani Iron Mines of the Tata Iron & Steel Co., at Gorumahisani, India.

**Paul Stuart Rattle, Salt Lake City, Utah.**

Proposed by L. A. Jeffs, W. A. Wilson, W. C. Ralston, Stuart Hazlewood.

Born 1882, Oak Park, Ill. 1905, M. E., Cornell University. 1901, Graduated from Lewis Institute, Chicago, Ill. 1888-96, Common Schools, Oak Park, Ill. 1905-06, Standard Steel Car Co. 1906-07, Dayton Hydraulic Machinery Co. 1907-08, Dist. Mgr., Dayton Hyd. Mach. Co., Chicago. 1909, Mech. Engr., B. M. Osburn Co. 1910, Hicks Locomotive & Car Works. 1910-14, Mech. Engr., Mine & Smelter Supply Co. (Denver). 1914, Midvale Steel Co., Salt Lake City, Utah.

Present position: Sales Dept., Midvale Steel Co., Cambria Steel Co., Worth Bros. Co., all of Salt Lake City, Utah.

### *Junior Members*

**Joseph P. Ruth, Jr., Denver, Colo.**

Proposed by Fred Carroll, A. S. Blomfield, H. S. Coe.

Born 1894, Denver, Colo. 1910-13, North Denver High School. 1916-17, Y. M. C. A. Preparatory School. Will attend Colo. School of Mines in Sept., 1917. 1915-16, Vindicator Cons. Gold Min. & Mill. Co. and Portland Gold Min. & Mill. Co. All work with mechanical and experimental flotation, i. e., designing flotation machines.

Present position: Student.

**Clarence Theodore Todd, Butte, Mont.**

Proposed by J. T. Shimmin, S. E. Hollister, A. W. Hackwood.

Born 1891, Los Angeles, Cal. 1907-11, Los Angeles Polytechnic H. S. (Graduated). 1911-14, Colorado School of Mines, Golden, Colo. 1914, Mining, Vindicator Mine, Cripple Creek, Colo. 1914, Mine Sampling at Smuggler Union Mine, Telluride, Colo. 1914-15, Constructing and operating blast furnace at Wanokah Mining Co.'s plant, Ouray, Colo. 1915-16, Head of Solution Dept., Vanadium Mill, Vanadium, Colo.

Present position—1916-17: Flotation operator, Butte & Superior Mill.

Més Edmund Waechter, Cleveland, Ohio.

Proposed by C. E. Locke, E. E. Bugbee, C. R. Hayward.

Born 1893, Iványos, Hungary. 1913-16, Case School of Applied Science. 1916-17, Mass. Institute of Technology.

Present position: Student.

### CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period June 10, 1917, to July 10, 1917.

This list together with the list published in Bulletin Nos. 121 to 127, January to July, 1917, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1917, and brings it up to the date of July 10, 1917.

|                            |   |
|----------------------------|---|
| ADKINS, HARVEY S.,         | Supt., J. B. Elkhorn Coal Co., Esco, Pike Co., Ky.                                  |
| ALLEN, MILTON A.,          | Arizona State Bureau of Mines, Tucson, Ariz.  |
| ARMSTEAD, HENRY H.,        | Blacktail, Idaho on Lake Pend d'Oreille.  |
| BAHNEY, L. W.,             | Met. Engr., Scovill Manufacturing Co., Waterbury, Conn.                             |
| BARBOUR, PERCY E.,         | Deputy Supt., New York State Troopers, Capitol Bldg., Albany, N. Y.                 |
| BEHR, H. C.,               | Cons. Mech. Engr., Room 827, 2 Rector St., New York, N. Y.                          |
| BEHRE, HENRY A.,           | 840 Park Ave., New York, N. Y.  |
| BERG, HAAKON A.,           | 143 W. Swissvale Ave., Swissvale Branch, Pittsburgh, Pa.                            |
| BLACK, HERBERT F.,         | Asst. to President, The Midvale Steel Co., and Cambria Steel Co., Pittsburgh, Pa.   |
| BLOCK, JAMES A.,           | Instructed to hold all mail.  |
| BORDEAUX, ALBERT F. J.,    | S. Real by S. Pierre d'Albigny, Savoy, France.                                      |
| BOUERY, PIERRE,            | Brioude, H de Loire, France.  |
| BOYER, SAMUEL L.,          | 111 Carl St., San Francisco, Cal.   |
| BRIGGS, CHARLES H.,        | Bonne Terre, Mo.  |
| BRINGS, HENRI,             | Goldfield, Nev.   |
| BROWN, GILMOUR E.,         | Care Messrs. George McBain, No. 1, The Bund, Shanghai, China.                       |
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 WILEY, HOWARD . . . . . Magma Copper Co., Magma, Ariz.  
 WRAMPFELMEIER, ERNEST L. S. . . . . Ruby, Ariz.  
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| Name.  | Last address of Record from which Mail has been returned.   |
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| BORDWELL, EDWARD M.                                | Masonic, Cal.   |
| CASTLETON, WILLIAM A.                              | Canyon Ferry, Mont.   |
| DENNIS, ARTHUR C.                                  | Gulf Production Co., 915 North Dixon St., Gainesville, Tex. |
| DENNIS, PAUL J.                                    | Gen'l Delivery, Denver, Colo.                               |
| HOOKEK, WALTER, 75 Parliament Hill Mansions,       | Gospel Oak, Highgate N., London, England.                   |
| KERR, DUNCAN M.                                    | Winthrop, Cal.  |
| KLEINSCHMIDT, FRANZ H., Blue Jacket & Queen Mines, | Landore, Adams Co., Idaho.                                  |
| LOWE, HENRY P.                                     | Central City, Colo.   |
| RAY, JAMES C.                                      | Oatman, Ariz.   |
| RECKNAGEL, RUDOLF.                                 | 13 Austin Friars, London, E. C., England.                   |
| REID, JOSEPH W.                                    | El Rito, New Mex.   |
| SCHNEPP, CHARLES F.                                | Riverside Club, Penns Grove, N. J.                          |
| SMITH, ELWYN L.                                    | Ray, Ariz.  |
| TONG, SING K.                                      | 404 West 115th St., New York, N. Y.                         |

# NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period June 10 to July 10, 1917.

| Date of Election. | Name.              | Date of Death.  |
|-------------------|--------------------|-----------------|
| 1905              | Arnold, Leo F.     | June 1, 1917.   |
| 1895              | Grace, William F.  | Jan. 13, 1917.  |
| 1874              | Hague, Arnold.     | May 14, 1917.   |
| 1915              | Hall, William T.   | May 19, 1917.   |
| 1908              | Pryor, James.      | April — 1917.   |
| 1911              | Radcliffe, Alfred. | April 22, 1917. |

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 30th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Pyritic Deposits Near Røros, Norway

BY H. RIES,\* PH. D., AND R. E. SOMERS,† PH. D., ITHACA, N. Y.

(St. Louis Meeting, October, 1917)

### Introduction

BODIES of pyritic ore in schistose rocks have long been known in different parts of the world. The several occurrences resemble each other in being usually of more or less lenticular shape, inclosed in walls of schist or gneiss, and carrying pyrite, chalcopyrite, and pyrrhotite in varying proportions, as the chief ore minerals. The deposits differ, however, in that sometimes one, sometimes another, of the three sulphides mentioned may predominate. The orebodies in general are more or less closely conformable with the schistosity of the wall rocks, although in cases this may not hold true. The boundaries may or may not be sharp, but in general are fairly distinct. The lenses vary in size, may occur singly, or in groups; in the latter case they may be in line, overlapping, or occasionally parallel. Pinching and swelling of individual lenses is not uncommon.

The genesis of these pyritic bodies has provoked much discussion, and they have been variously classed as sedimentary, igneous intrusions, hydrothermal replacements or impregnations, etc. Indeed, in some cases different geologists have most positively assigned widely diverse origins to the same deposit, as in the case of Rio Tinto<sup>1</sup> and Rammelsberg.<sup>2</sup> It is true that in the light of modern criteria, we have been able in a few cases to decipher clearly the process of origin and so there seems no doubt now regarding the genesis of the deposits at Rio Tinto, Spain;<sup>3</sup> Rammelsberg, Germany;<sup>4</sup> Ducktown, Tenn.;<sup>5</sup> southwestern Virginia;<sup>6</sup>

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<sup>1</sup> A. M. Finlayson: *Economic Geology* (1910), 5, 357.

<sup>2</sup> J. H. L. Vogt: *Zeitschrift für praktische Geologie* (1894), 173; W. Lindgren and J. D. Irving: *Economic Geology* (1911), 6, 303.

<sup>3</sup> Finlayson: *Loc. cit.*

<sup>4</sup> Lindgren and Irving: *Loc. cit.*

<sup>5</sup> W. H. Emmons and F. B. Laney: U. S. Geological Survey, *Bulletin* 470 (1911), 151.

<sup>6</sup> W. H. Weed and T. L. Watson: *Economic Geology* (1906), 1, 309.

eastern Quebec;<sup>7</sup> and Kyshtim, Russia.<sup>8</sup> The third and fourth are predominantly pyrrhotite, the others mostly pyrite. There are others, however, regarding which we may be supposed to lack positive proof, such as Bodenmais, Bavaria;<sup>9</sup> and Saint Bel, France;<sup>10</sup> while even the great series of Norwegian occurrences are still a subject of dispute.

The present paper deals with certain of these occurrences which are being worked near Röros, Norway, and from which a suite of specimens was collected by the senior author during a visit to that locality in the summer of 1914, the results derived from the study of which it is hoped may throw some further light on the problem of their origin.

The Norwegian pyritic deposits are widely distributed.<sup>11</sup> In the northern part of Norway they are known around Birtavarre, Sulitjelma, Bossmo, etc.; in central Norway around Ytterö, Meraker, Meldalen, Killingdal, Kjöli, Röros, Röstvangen, Foldal, etc.; in southern Norway in Söndhordland and Vignäs. The deposits are said to occur at various horizons of the Cambro-Silurian, and in every case in areas of regional metamorphism. These metamorphic rocks are part of an uplifted area, which extends from the British Isles to northern Norway, and was raised at the end of the Silurian to form the Norwegian mountains. The metamorphosed Paleozoics include gray to green phyllites, conglomerates, sandstones, clay slates and limestones, but the wall rock of the ores in most cases is phyllite. The intrusive rocks found within the area range from soda granites to gabbros or peridotites (Vogt), and some of the igneous rocks, such as the granulites, seem to have been intruded during the period of folding.<sup>12</sup> The igneous rocks may be altered, the gabbro especially being changed to the saussurite type.

The orebodies are practically all flat lenses, of varying dip, occurring usually between schist walls, but sometimes within shear zones in the gabbro. While the lenses conform in a general way to the schistosity of the metamorphic rock, they may sometimes cut across it, or even send out stringers across the foliation. Moreover, in some occurrences a breccia of ore and wall-rock fragments has been noticed. A noteworthy feature is that in nearly every case of ore in phyllite, a gabbro mass is not far away.

### *The Röros Deposits*

The mines visited were the Sextus (Fig. 2) and Kongens (Fig. 1), situated about 8 miles (12.88 km.) west of north from Röros. In addi-

<sup>7</sup> J. A. Bancroft: *Dept. Col'n. Mines and Fisheries, Mines Branch, Quebec*, 1915.

<sup>8</sup> A. W. Stickney: *Economic Geology* (1915), 10, 593.

<sup>9</sup> E. Weinschenk: *Zeitschrift für praktische Geologie* (1900), 65.

<sup>10</sup> deLaunay: *Ibid.* (1901), 161.

<sup>11</sup> Vogt, Krusch and Beyschlag: *Ore Deposits* (Translation), 1, 304.

<sup>12</sup> O. Falkenberg: *Zeitschrift für praktische Geologie* (1914).

tion, specimens were also obtained from the Storvarts mine, although this was not visited. The rocks and ores described come mainly from the two first-named mines.



FIG. 1.—VIEW ALONG OUTCROP OF KONGENS OREBODY, SHOWING CUT FROM WHICH ORE WAS REMOVED, GABBRO WALL ON RIGHT AND PHYLLITE WALL ON LEFT.

#### *Associated Rocks*

*Schist.*—The schist associated with the orebodies at the Sextus and Kongens mines is of a greenish-gray color and fine texture. While usually thinly foliated because of a predominance of chlorite, it becomes in



FIG. 2.—VIEW LOOKING TOWARD HILL IN WHICH SEXTUS OREBODY LIES.

places more massive, owing to greater abundance of quartz. One specimen showed small needle-shaped crystals of actinolite. Some of the schist shows strong crumpling, and also lenticular quartz veins, which partake of the folding.

Under the microscope (Fig. 3), the schist is seen to consist usually of an intergrowth of fine-grained quartz, chlorite, brown and green biotite, and

sometimes actinolite, the bladed minerals showing a parallel arrangement. Epidote and plagioclase are not uncommon, while zoisite, apatite, rutile, magnetite and pyrite were noted. There seems no doubt that chlorite is a normal constituent of the rock and not introduced by hydrothermal action preceding ore deposition, because of its manner of intergrowth with the other common silicates of the schist.

*Saussurite-gabbro*.—No gabbro is found in contact with the ore at the Sextus mine, but at the Kongens a mass of the intrusive lies not far from the sulphides, and, as mentioned later, is in actual contact with them at one place.



FIG. 3.—THIN SECTION OF CHLORITE, BIOTITE, QUARTZ SCHIST. ORDINARY LIGHT.  $\times 26$ .



FIG. 4.—THIN SECTION OF SAUSSURITIZED GABBRO, SHOWING POIKILITIC INTERGROWTH OF LARGE LATH-SHAPED AMPHIBOLES WITH QUARTZ, IN FINER-GRAINED GROUNDMASS. ORDINARY LIGHT.  $\times 26$ .

Macroscopically the saussurite-gabbro is a greenish-gray, dense, fine-grained rock, showing hornblende needles of varying size, the largest being  $\frac{3}{4}$  in. long and  $\frac{1}{8}$  in. wide, while the smallest were scarcely noticeable. Under the microscope (Fig. 4), the gabbro exhibits prominent lath-shaped crystals of a blue-green amphibole, imbedded in a fine groundmass composed of quartz with lesser amounts of chlorite, epidote, biotite and magnetite. Small rounded grains of quartz occur in the amphibole as poikilitic intergrowths.

### Ore

Vogt in an earlier paper on the Norwegian pyritic<sup>13</sup> deposits, recognized three types of ore, viz.: (1) Chiefly pyrite, usually with some chalc-

<sup>13</sup> *Zeitschrift für praktische Geologie* (1894), 41.

pyrite and blende, but generally with pyrrhotite and only a trace of galena; (2) pyrite and chalcopyrite, with comparatively much quartz and hornblende; and (3) pyrite with chalcopyrite, quartz, etc. Falkenberg,<sup>14</sup> on the other hand, says that in the Røros district, particularly, the ores are: (1) Copper-bearing pyrite and (2) copper-bearing pyrrhotite.

Such a grouping, it seems to the authors, is likely to create the impression that these ore types are more sharply defined than they really are. In the Sextus and Kongens mines, pyrite, chalcopyrite or even pyrrhotite may be the predominating sulphide in one or another part of the orebody, but the several types appear to grade into each other, and do not seem to be confined to any particular mine or locality.

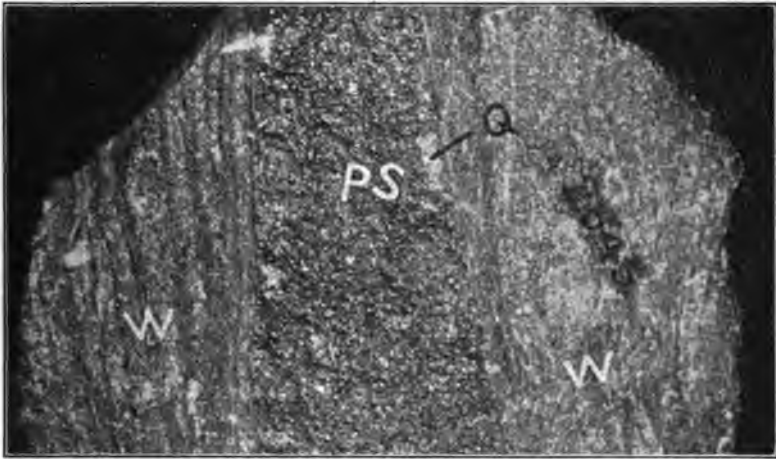


FIG. 5.—HAND SPECIMEN OF QUARTZOSE SCHIST, WITH BAND OF PYRITE AND SPHALERITE (PS), SHARPLY SEPARATE FROM WALL ROCK (W). NOTE SMALL LENS OF LATER QUARTZ (Q).  $\times \frac{3}{4}$ .

At both the Kongens and Sextus mines, the ore forms low dipping lenses enclosed in schist, and at the latter mine, there was a second or upper lens which had been worked out. The gabbro lies above the ore, and is not in contact with it, except in the Kongens mine, where, at one place on the outcrop, one wall of the cut is schist and the other is gabbro (Fig. 1). The ore here is said to have been in actual contact with the intrusive, but as the latter showed evidence of strong folding, and the surface of it was slickensided, the contact may have been a structural rather than an intrusive one. The ore lenses which are usually conformable with the phyllite, pinch and swell, and at times inclusions of the country rock are found in the ore. Quartz lenses are also scattered through it, but are probably of later origin.

The ore boundaries are fairly sharp, and bands of sulphides may

<sup>14</sup> *Loc. cit.*

occur in the wall rock. These are conformable to the schistosity, and also show a fairly sharp boundary (Fig. 5).

Some specimens from the Sextus mine show an irregular mixture of sulphides and crumpled schist, the former being chalcopyrite, which appears to have been injected prior to the crumpling, as the sulphide shows slickensiding. At one place in the same mine, a fracture about  $\frac{5}{8}$  in. in thickness was observed to cut the orebody from roof to floor. It was lined with small pyrite cubes, on top of which were larger pyritohedrons, which were in turn coated with calcite. It seems not unlikely that this fissure represents a shrinkage crack, on whose walls the small pyrite cubes crystallized at the same time as the sulphides in the orebody. The



FIG. 6.—POLISHED ORE SPECIMEN, WITH PREDOMINANT PYRITE SHOWING CRYSTAL BOUNDARIES, AND SPACES FILLED IN BY SPHALERITE (S) AND CHALCOPYRITE (Cp).  $\times 22.5$ . A COMMON TYPE OF ORE.



FIG. 7.—POLISHED ORE SPECIMEN SHOWING RESORBED PYRITE (P), SURROUNDED BY CHALCOPYRITE (Cp) AND SPHALERITE (S).  $\times 22.5$ . A COMMON ORE TYPE.

pyritohedrons were probably deposited later by circulating waters. The texture of the ore ranges from fine-grained, granular, massive pyrite, to coarse-grained ore which consists of pyrite crystals, embedded in a ground-mass of what macroscopically seems to be chiefly massive chalcopyrite.

In order to study more closely the minerals in the ore, as well as their relation to the wall rock, a number of thin sections and polished specimens were prepared.

Polished surfaces of the massive ore, examined by reflected light, show pyrite, chalcopyrite, pyrrhotite and sphalerite as the metallic minerals.<sup>15</sup> The pyrite is undoubtedly the oldest. It occurs in the form of individual grains, which are bounded by crystal faces where the pyrite is abundant (Fig. 6), and yet had room to grow, but show a rounded

<sup>15</sup> Magnetite, arsenopyrite and galena reported by other observers from the Norwegian deposits were not found in any of the specimens examined by us.



outline (Figs. 7 and 8) due to resorption when the other sulphides are greatly in excess of their eutectic with pyrite. The three other sulphides are younger than the pyrite, but in most cases do not show any well-



FIG. 8.—POLISHED ORE SPECIMEN SHOWING RESORBED PYRITE (P), IN GROUNDMASS OF CHALCOPYRITE (Cp) WITH SOME PYRRHOTITE (Pr).  $\times 22.5$ .



FIG. 9.—POLISHED ORE SPECIMEN SHOWING FINE-GRAINED INTERGROWTH OF CHALCOPYRITE (Cp.), SPHALERITE (S), AND PYRRHOTITE (Pr).  $\times 22.5$ .

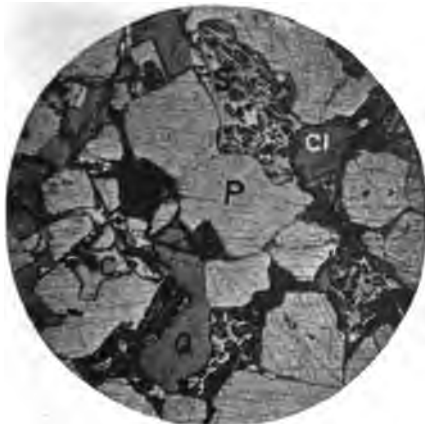


FIG. 10.—POLISHED ORE SPECIMEN SHOWING PYRITE (P), CORRODED BY CHALCOPYRITE (Cp) AND SPHALERITE (S); (Cl), CHLORITE, (Q), QUARTZ.  $\times 22.5$ .



FIG. 11.—POLISHED ORE SPECIMEN SHOWING FRACTURED PYRITE (P), WITH THE CRACKS FILLED WITH SPHALERITE, CHALCOPYRITE, AND PYRRHOTITE. SURROUNDING MATERIAL MOSTLY PYRRHOTITE WITH SOME CHALCOPYRITE.  $\times 22.5$ .

determined order of crystallization among themselves, the boundaries being more or less mutual.<sup>16</sup> The sphalerite and chalcopyrite probably

<sup>16</sup> Falkenberg gives the order of formation as pyrite, chalcopyrite, pyrrhotite, sphalerite, magnetite, bornite, chalcocite, galena. Stutzer, however, believes that arsenopyrite and magnetite precede the pyrite.

began to separate out before all the pyrite had crystallized, because it contains inclusions of them, but the pyrrhotite does not appear to have begun to crystallize so early, and in one specimen inclusions of sphalerite and chalcopyrite were found in it. Inclusions of pyrrhotite in pyrite are extremely rare. At times the three later sulphides show a rather fine-grained even intergrowth (Fig. 9), but otherwise the chalcopyrite always forms larger areas. Of these three the chalcopyrite is the most abundant, with sphalerite next and pyrrhotite last. Occasionally, however, the pyrrhotite may form most of the ore.

The later formation of the chalcopyrite, sphalerite and pyrrhotite is clearly shown in two ways, *i.e.*: (1) by the corrosion of the pyrite by the other sulphides (Fig. 10) and (2) by the entrance of these sulphides along



FIG. 12.—THIN SECTION OF LEAN ORE SHOWING SULPHIDES (S), REPLACING CHLORITE (C) AND PENETRATING ALONG ITS CLEAVAGE PLANES.  $\times 26$ . THIS IS CUT FROM A SPECIMEN OF CRUMPLED SCHIST FROM MARGIN OF SEXTUS OREBODY.



FIG. 13.—POLISHED ORE SPECIMEN SHOWING INCLUDED FRAGMENT OF SCHIST (G), PENETRATED BY CHALCOPYRITE (Cp), SPHALERITE (S) AND PYRITE (P). SURROUNDING GROUNDMASS IS OF CHALCOPYRITE WITH SPHALERITE AND PYRRHOTITE.  $\times 22.5$ .

fractures in the pyrite (Fig. 11). Where pyrite is very abundant, little or no corrosion has occurred, but where the other sulphides predominate this feature is pronounced (Fig. 8). It consists in the formation of embayments in some, and the rounding of corners of the grains in others.

A fracturing of the pyrite has in some cases occurred before the other sulphides crystallized, because they fill the cracks, but the fissuring seems to have continued subsequent to the solidification of all the sulphides.

Included fragments of the schist, as well as the schist walls of the orebody, are replaced by the sulphides (Fig. 12), and also penetrated along cleavage planes by tongues of the metallic minerals. Of the several sulphides present, the chalcopyrite shows a much greater tendency to send

out stringers along the cleavage planes of the chlorite (Fig. 12), than the others. The schist fragments within the ore are not oriented, and, moreover, they are often bent, twisted and shredded (Fig. 13).

Quartz grains are found in the ore. They at times seem to be contemporaneous with the sulphides, but in most cases they are genetically related to the chlorite and other minerals of the schist and seem to be of similar age. The former relationship of quartz to sulphides was also noticed by Falkenberg in the deposits at Lokken.<sup>17</sup> But small lenses or eyes of quartz were also found (Fig. 5).

Some ore specimens show an indistinct banding due to the predominance of one of the later sulphides along certain lines, and corrosion of



FIG. 14.—THIN SECTION OF LEAN ORE FROM EDGE OF SEXTUS OREBODY SHOWING SULPHIDES (S) REPLACING CHLORITE (C).  $\times 26$ .

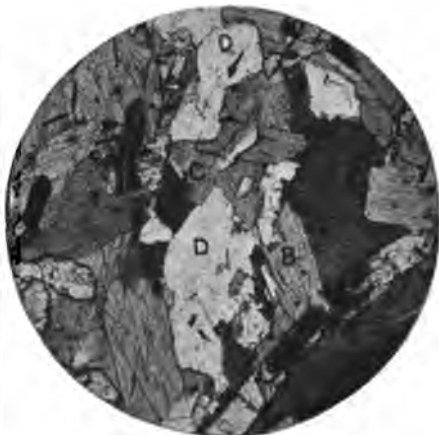


FIG. 15.—THIN SECTION OF WALL ROCK NEAR ORE IN STORVARTS MINE, SHOWING DOLOMITE (D) REPLACING CHLORITE (C) AND BIOTITE (B); ALSO SULPHIDES (BLACK) REPLACING DOLOMITE AND CHLORITE.  $\times 26$ .

the pyrite is more noticeable in those bands in which sphalerite is more abundant.

#### *Relationship of Sulphides to Non-Metallics*

It has already been pointed out in describing the schist country rock that the chlorite, biotite, garnet, feldspar and quartz, were contemporaneous. Study of thin sections shows that the schist may contain veinlets of later dolomite (Fig. 15) and quartz, both of which minerals can be seen replacing at least the chlorite (Fig. 15). Later than any of these are the ore minerals, and their relationship to the silicates is extremely interesting, where it can be studied. This, of course, is chiefly along the boundary of the orebody where it comes into contact with the wall rock

<sup>17</sup> *Zeitschrift für praktische Geologie* (1914).

(Figs. 12, 14–17). To the naked eye, the contact between the two appears quite sharp, but even under low powers the polished rock shows the sulphides extending irregularly and for a short distance into the schist. The relationship between sulphides and schist can also be studied where the former surrounds and penetrates inclusions of the latter (Fig. 13). In some cases the sulphides form films between the cleavage plates of the chlorite, but at others they can also be distinctly seen replacing minerals of the schist such as chlorite (Fig. 16), biotite, garnet (Fig. 17), dolomite (Fig. 15) and quartz.

Careful search of all the thin sections failed to reveal any sericite, and in this connection it is interesting to note that all the feldspar grains



FIG. 16.—THIN SECTION OF LEAN ORE FROM NEAR MARGIN OF KONGENS ORE-BODY, SHOWING SULPHIDES (S) REPLACING QUARTZ (Q) AND CHLORITE (C).  $\times 26$ .

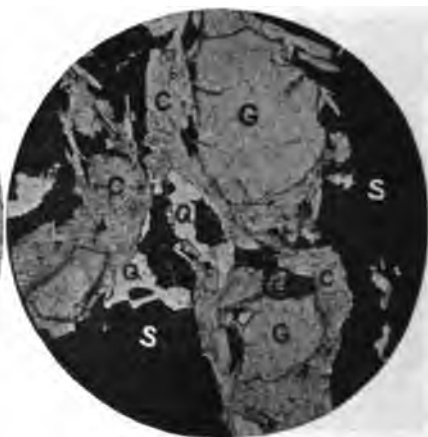


FIG. 17.—THIN SECTION OF WALL ROCK CLOSE TO ORE (SEXTUS MINE) SHOWING SULPHIDES (BLACK) REPLACING GARNET (G), CHLORITE (C) AND QUARTZ (Q).  $\times 26$ .

observed were unaltered. Brown biotite is sometimes found in small grains contemporaneously intergrown with the sulphides.

Another important point to be stated is that the presence of chlorite does not seem to be dependent on the processes of ore introduction, since it is found in both the mineralized and unmineralized schist in its normal relationship to quartz and the other silicates.

### *Origin*

*Previous Views.*—Many theories have already been expressed regarding the origin of these ores. The earlier writers seemed to lean strongly toward a sedimentary origin, including Helland<sup>18</sup> in the seventies and

<sup>18</sup> Vorkommen von Kiese in gewissen Schiefen in Norwegen, *Universitäts Programm*, 1873.

later in Germany, Stelzner,<sup>19</sup> Klockman,<sup>20</sup> as well as others. They based their theory on the apparently uniform concordance of the orebodies with the foliation of the schist, the fact that orebodies and schist had (they thought) been folded together, and apparent banding in the ore. The lens-shaped nature of the orebodies was explained by some as due to their being beach deposits. Vogt also at one time regarded the ores as chemical precipitates derived from submarine exhalations emanating from the gabbro. Reusch suggested the possibility of the sulphides having replaced limestone, but there is no field evidence to bear this out. Sjögren,<sup>21</sup> after studying Sulitelma, advanced the somewhat curious view that iron and copper solutions derived from the weathering of pyrite in the gabbro, and containing also alkali carbonates from the same source, had percolated downward and come into contact with rising waters carrying hydrogen sulphide, resulting in a precipitation of the iron and copper sulphides. He even suggested that this process might apply to other Norwegian occurrences.

At the present day, most geologists no doubt believe that the Norwegian pyrite deposits like those of the Røros and Sulitelma type are of epigenetic character. This theory was early expressed by Kjerulf,<sup>22</sup> and the earlier French investigators<sup>23</sup> Durocher and Duchanoy, who believed that we must regard the pyrite as intruded, and that the ore followed the planes of schistosity because it was the path of least resistance.

Stutzer,<sup>24</sup> after studying Sulitelma, came to regard the deposits as *injection veins (injections-gänge)*, and in this connection pointed out that there may be every gradation between a watery solution containing dissolved minerals from a magma, and a magma containing a comparatively small amount of water. He furthermore drew attention to the fact that during the crystallization of a magma the metallic minerals may separate and segregate at practically the same time as the silicates, or they may segregate and crystallize out a little later, but before doing the latter, wander into the already crystallized part of the magma or even the wall rock, giving what Sjögren named *diamagmatic ores*. He therefore regards the sulphide bodies like those of Sulitelma and Røros as of this class.

Falkenberg,<sup>25</sup> after a somewhat extensive study of the deposits of

<sup>19</sup> *Die Sulitelma Gruben in nordlichen Norwegen*, Freiberg, 1891.

<sup>20</sup> *Zeitschrift der Deutschen Geologischen Gesellschaft* (1893), 45, 281.

<sup>21</sup> *Geol. Fören. Förrh*, November, 1893.

<sup>22</sup> *Geologie des südlichen und mitteren Norwegen*, 1880.

<sup>23</sup> *Annales des Mines* (1849), ser. 4, 15, 171, 267 and (1854), ser. 5, 5, 181.

<sup>24</sup> *Zeitschrift für praktische Geologie* (1909), 17, 130, and *Oesterreichs Zeitschrift für Berg. und Hüttenwesen* (1906), No. 44, 567.

<sup>25</sup> *Zeitschrift für praktische Geologie* (1914), 22.

southern Norway, evidently considers many of them to be of eruptive origin. For he says that the common forking of many of the sulphide orebodies, which is closely associated with brecciation, can be explained only on eruptive grounds. The corrosion phenomena (also noted by us) he believes explainable only by crystallization from fusion. After commenting on the close association of the orebodies and gabbro, he remarks that he considers it difficult to state definitely the nature of the magma, but believes that no doubt water and a small amount of silica reduced the viscosity of the molten or water solutions of high temperature, thus permitting them to ascend the fissures of the surrounding rock.

Lindgren<sup>26</sup> expresses the view that some deposits of solid pyritic minerals present features which can hardly be explained otherwise than by actual injection of molten sulphides, perhaps to be considered as residual solutions from adjoining intrusive bodies.

Vogt, Krusch and Beyschlag,<sup>27</sup> after emphasizing the close relationship of most of the sulphide orebodies with the gabbro, make the statement that the formation of the pyrite bodies is contemporaneous with the magmatic period of the eruptive rocks, and that it follows from the form and structure of the deposits that they could not have been formed by long-continued deposition from different solutions, but that the whole of the material must have been introduced or injected at once. Furthermore, that these pyrite deposits are products of magmatic differentiation. They agree with Brögger in believing that the rounded idiomorphic pyrite crystals not only show resorption, but conclude that this was of magmatic character.

*Authors' Theory.*—The theory of a sedimentary origin hardly seems any longer tenable, and we must therefore recognize that these orebodies are of later age than the enclosing rocks. It still remains, however, to be explained whether they are veins or dikes, a problem by no means easy, because of the lack of well-defined and never-failing criteria for identifying these two processes.

If the orebodies are veins, one might expect to find some good evidence of hydrothermal alteration of the walls, such as the development of chlorite or sericite. It is true that chlorite is found in the rock associated with the sulphides, and even replaced by them, but its relationships there are so identical with those in the unmineralized schist, that we can see no good reason for regarding it as secondary, in the sense that it is a product of vein-wall alteration, and not of regional metamorphism. The absence of sericite is also significant, for there is feldspar present which could have yielded the necessary elements for its development. These facts, therefore, would seem to eliminate one of the most strongly held criteria for vein formation by deposition from solution.

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<sup>26</sup> *Mineral Deposits*, 768.

<sup>27</sup> *Ore Deposits*, 1, 312.

In view of the fact that the ore contains at least two minerals (pyrrhotite and biotite) which are characteristic of high-temperature conditions, it hardly seems likely that these materials could have been introduced in hot aqueous solutions without some hydrothermal alteration of the walls.

The alternative view would therefore be to regard these orebodies as dikes. This idea, as applied to the formation of sulphides, has perhaps been regarded as a somewhat radical one, but in our opinion is not at such great variance with the commonly accepted view of vein formation by precipitation from solution. Any magma is a natural mixed solution of rather complex character. It may differentiate, and the lighter constituents tend to rise, while the heavier sink either in a subcrustal reservoir or in an intrusive body such as a sill. These lighter constituents include the water and other vein-forming compounds, which escape from the magma during cooling, and may form veins in the surrounding rocks. Portions of the remaining magma may make their way into fissures and form dikes of either acid or basic character, and if very basic they might be composed almost entirely of sulphides or metallic oxides.<sup>23</sup> In any case, however, it is a deposit from solution, and essentially the difference lies in the materials present.

We have come to say that a solution with much water in it deposits a vein. It comes from the top of the magmatic unit,<sup>22</sup> and contains other mineralizers also, which alter the country rock. On the other hand, there may be a solution that gets a favorable chance to leave the very bottom of the unit, and be composed almost wholly of metallic compounds. It does not alter the country rock, because most of the silica, water, etc., are elsewhere. But it may form a fissure deposit, solidify according to the laws of solutions, and replace walls or inclusions with its own minerals. Even such a basic mass as a mixture of sulphides might contain a little water or other mineralizers, and this may be borne out by the finding of small and scattered amounts of such minerals as tourmaline, apatite, cordierite, etc., by other investigators, although one cannot always be sure from their descriptions whether they are ingredients of the metamorphic rocks, developed independently of ore-forming processes.

There seems to us then no serious objection to the theory that the deposits of R  ros and similar Norwegian ones have originated as outlined above, and represent injections of magmatic sulphides coming probably from the same reservoir as the gabbro.

*Criteria to be Applied.*—Having expressed our opinion regarding the origin of these orebodies, it may be well to consider in a little more detail some of the criteria by which such deposits are to be judged. It has been stated that so far as the R  ros occurrences mentioned are concerned

<sup>22</sup> Iron Mountain, Wyoming; or Kiruna, Sweden.

<sup>23</sup> B. S. Butler: *Economic Geology* (1915), 10, 101.

there is no evidence of hydrothermal action on the walls, and this is at least negative evidence in favor of the view expressed. Since the ore-bodies do not represent magmatic segregation and crystallization in place, we cannot look for such criteria as primary intergrowths with silicates of the original magma. Corrosion of pyrite by other sulphides, though regarded by some as evidence of molten conditions, is not necessarily such, since the same may be observed in pyritic deposits formed by hydrothermal replacement. Fig. 18 shows a case of this from the King mine at Ascot, Que., photographed from a polished specimen kindly supplied us by Prof. J. A. Bancroft of McGill University. Here the pyrite and sphalerite have been deposited by replacement of schist, the sphalerite, however, having corroded the pyrite. A point to be considered



FIG. 18.—POLISHED ORE SPECIMEN FROM ASCOT, QUEBEC. SHOWS PYRITE (P) CORRODED BY SPHALERITE (S), IN GROUNDMASS OF CHALCOPYRITE (CP), SPHALERITE, AND QUARTZ (Q).  $\times 22.5$ .

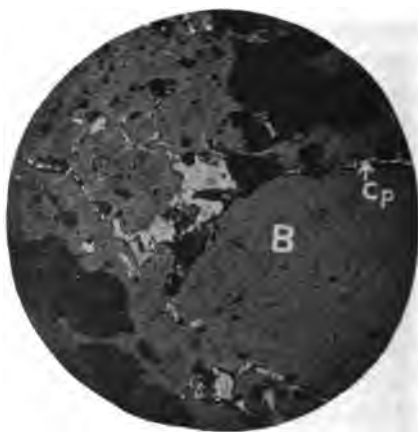


FIG. 19.—POLISHED SURFACE OF FIRE BRICK (B), SHOWING PENETRATION OF MOLTEN SULPHIDE (CHALCOPYRITE (CP) ALONG MINUTE CRACKS.  $\times 22.5$ . THE VEINLET INDICATED BY ARROW IS ABOUT  $\frac{1}{50}$  MM. THICK.

is the entrance of molten sulphides into the schist. This has been largely parallel with the schistosity, and it is possible that the sulphide intrusion had sufficient force behind it to widen the fracture it followed.

So far as the mines visited at Røros are concerned, there were few observed inclusions of wall rock in the ore, although at several other localities they have been noted, and in some cases a breccia of rock fragments in a matrix of ore has been mentioned. In our polished specimens, detached grains of schist were not infrequently observed (Fig. 13), and often in such positions as to indicate that they had been pried off the wall. This fact was also noted by Stutzer, and some of the illustrations given by Vogt produce the same impression. Such scattered and un-oriented fragments would be in accordance with the injection theory.

Both these fragments (Fig. 13), and also the schist of the wall itself,



showed minute filaments of sulphide, entering between the cleavage flakes, of the chlorite, and it might be thought that molten sulphides could hardly be sufficiently fluid to penetrate the rock in such thin filaments. This, however, is by no means impossible, for Stutzer has described the penetration of gneiss by veinlets of molten sulphide,<sup>20</sup> and in Fig. 19 we give a photomicrograph of a firebrick from the hearth of a furnace at Trail, B. C., which is quite extensively impregnated with chalcopyrite, bornite and chalcocite. The chalcopyrite stringer indicated by the arrow in figure has a diameter of 0.02 mm.<sup>21</sup>

The lenticular character of the orebodies needs some consideration, and we are not able to state whether it is altogether an original structure. Some features point to its being due to post mineral movement, which caused pinching of the orebodies. That such pressure has probably been exerted is shown by the minute fractures in the pyrite; slickensided surfaces in some of the ore specimens; the folding and smoothing of the gabbro at the outcrop of the Kongens orebody; and lastly, the occurrence of corroded pyrite grains in schist fragments, near the border of the orebody, their relation being such as to suggest that they had possibly been pushed into the schist.

The massive and uniform character of the ore, freedom from bands and grains of wall-rock minerals, while suggesting igneous injection, is not absolute proof of it, for complete replacement may give equally massive ore, and even coarsely granular material, so that there seem to the authors no means of telling from the hand specimens of the material whether or not it was replacement, ordinary vein filling, or igneous injection. In Fig. 20 we give a photomicrograph of a polished specimen of pyrite ore from Louisa County, Virginia, which is a replacement of schist or limestone, and yet so far as its appearance goes it does not differ essentially from some of the Røros material (Figs. 6 and 10).

That the orebodies were formed under conditions of high temperature is indicated by the presence of such minerals as pyrrhotite and biotite, and there is no doubt of the latter being primary, and not derived from pyrite by metamorphism (as claimed by Klockman), since the latter is not always in contact with the pyrite.

A final point to be mentioned is the mineral composition of the ore, and the objection to calling it magmatic injection because of the presence of pyrite.

Tolman and Rogers<sup>22</sup> in their valuable paper on magmatic sulphides, refer to the Norwegian pyrite deposits, but while they say that they are not personally familiar with them, do not hesitate to reject the view

<sup>20</sup> *Zeitschrift für praktische Geologie* (1908), 16, 119.

<sup>21</sup> See N. B. Davis: *Economic Geology* (1915), 10, 663.

<sup>22</sup> Leland Stanford Junior University Publications, 1916.

of their being magmatic sulphides. They doubt seriously whether pyrite is a characteristic magmatic mineral, and note that pyrrhotite seems to take the place of pyrite in magmatic ores, which they regard as but natural since pyrrhotite is the iron sulphide stable at high temperature." It is known, however, that pyrite may exist as a primary constituent of igneous rocks, and there seems no doubt that large pyrite bodies have been formed under high temperature conditions, like those of Louisa County, Virginia. These are but a step removed from injection masses, such as the Røros bodies may be, and because pyrite is of minor importance as a



FIG. 20.—POLISHED SURFACE OF ORE FROM ARMINIUS MINE, LOUISA COUNTY, VA., SHOWING PYRITE (P), SURROUNDED BY CHALCOPYRITE (Cp) AND SOME QUARTZ (Q).  $\times 22.5$ .

magmatic mineral in most orebodies of that type, it is not necessarily to be excluded from all.

### *Summary*

By way of summary, it would seem that the points favoring the injection theory are:

1. Absence of hydrothermal alteration of walls.
2. Uniformly close association with gabbro.
3. Unoriented inclusions of wall rock in ore.
4. Massive and sometimes porphyritic orebody, not in all cases absolutely conformable with enclosing schist.

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<sup>22</sup> E. T. Allen, J. L. Crenshaw and J. Johnston: The Mineral Sulphides of Iron, *American Journal of Science* (1912), **33**, 169.

**TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS**  
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**DISCUSSION OF THIS PAPER IS INVITED.** It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 29th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## **Comparative Tests of Hammer Drill Bits**

**BY CARROLL R. FORBES, E. M.,\* ROLLA, MO., AND JOSEPH C. BARTON, B. S.,  
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(St. Louis Meeting, October, 1917)

### **INTRODUCTION**

MANY different shapes of drill bits are in use with hammer drills, but little definite information is available whereby to judge which one of these shapes is the best.

The following investigation was undertaken in order to determine what effect, if any, the shape of the cutting edge has on the cutting speed and wearing qualities of drill bits.

The bits tested were those in common use, namely, the 4-point or cross bit, the 6-point bit, the Z bit and the "Carr" bit. The rock in which the tests were made was the red granite from southeast Missouri. It is a coarse crystalline granite containing an unusually large amount of quartz and is extremely hard, and on account of its uniform texture is admirably adapted to this work. While tests in other rocks might show somewhat different results, nevertheless it is the opinion of the authors that the relative cutting quality inherent in the shape of the bits would be the same in all rocks. Other qualities, such as mudding freely, freedom from fitching, etc., might make one bit more desirable than another in softer rocks.

### **DESCRIPTION OF BITS TESTED**

The 4-point bits were the usual shaped bits commonly made on the Leyner sharpener with  $14^{\circ}$  taper on the wings and a  $90^{\circ}$  angle between cutting edges. Some tests were run at 85 lb. pressure with a 4-point bit made with a  $5^{\circ}$  taper on the wings. This was made by using the Carr bit die.

The 6-point bits were the usual shaped bits made on the Leyner sharpener.

The Carr bits were made with a  $5^{\circ}$  taper and an angle of  $100^{\circ}$  between cutting edges.

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The Z bits were made with the regular Z-bit dolly which makes an angle of about  $60^\circ$  in the center and an angle of about  $45^\circ$  on the outer cutting edges. The center is made slightly higher than the outer edges. The Carr bit dies were used in forming Z bits to give them a  $5^\circ$  taper.

The Z-bit dolly was for solid steel, so it was necessary to drill out the center hole.

Several Z bits were tested at 85 lb. pressure, that were filed and hammered down, to give an angle of about  $100^\circ$  on the center edge and  $65^\circ$  on the outer edges.

The holes in the center of the bits were the ordinary size with the 4-point and 6-point and Carr bits. The hole in the Carr bit was much larger than those used in the others. A hole of the same size as in the Carr bits was used in the Z bit.



FIG. 1.

#### MANNER OF CONDUCTING TESTS

All tests were made with an Ingersoll-Rand "Jackhammer" drill. In the "down-hole" tests the drill was weighted with a 94-lb.

weight, thus insuring a constant pressure on the bit.

Most of the tests were made in vertical "up-holes," great care being taken to see that the drill was vertical. The tests consisted of drilling into a large block of granite supported on stringers over a concrete-lined pit as shown in the photograph (Fig. 2). For this work the drill was mounted on an air feed, which, however, was not large enough to produce the required pressure on the bit, so that it was necessary to counter-balance the weight of the machine by a bucket loaded with 40 lb. of scrap iron and connected to the machine by a rope over an overhead pulley.

Every effort was made to insure uniform conditions and to have the drill bit the only variable.

#### *Air Pressure*

A uniform air pressure was obtained by having a "pop-off" valve on the air receiver, which was connected by a 1-in. hose to the drill. The air compressor was regulated to supply sufficient air to keep the pop-off valve open and the drill running at the same time, the gage pressure remaining constant. The air receiver was drained after every third steel, as it was found in former work that this factor greatly influenced the quality of the air supply.

#### *Lubrication*

Uniform lubrication is most essential in work of this nature and was

insured by oiling the drill after each 1-min. test. The drill was taken apart and thoroughly cleaned and oiled after every third steel.

### *Steel*

The steel used was all of the same kind, F J A B  $\frac{7}{8}$ -in. hollow hexagon, and all comparative tests were run with steel of practically the same length. Lengths of 24, 48 and 62 in. were used.



FIG. 2.—METHOD OF DRILLING "UP" HOLES.

### *Sharpening and Tempering*

The greatest source of error in work of this kind is in the making and tempering of bits. It is impossible to make these conditions entirely uniform and many tests were made and the results discarded, because of non-uniformity in tempering. Any bit with a corner off, or which was chipped in any way (except the Z bit in a few cases), or which showed signs of being too hard or too soft, was rejected and resharpened until a perfect temper was obtained.

All bits were made and sharpened in a Leyner-Ingersoll 5A sharpener. The bits were heated, both for sharpening and tempering, in a gas muffle.

In sharpening and tempering the following rules were observed:

- (a) Heat to the usual working heat, taking care to see that the bit is heated uniformly throughout, and making as few heats as possible.
- (b) Heat only the end of the bit.
- (c) Never submerge a bit entirely under water until cold.
- (d) Keep shank end square.
- (e) Never let scale form by too much air.
- (f) When worked put in lime to anneal.
- (g) Heat the bit up to at least 800° and be sure to leave it in the furnace long enough to heat to the center; then draw from the fire and place in some dark chamber (tile, etc.) so the color can be observed, until the steel has cooled to about 800°; then dip into Sentinel paste which has melting point of 775° C., and let white residue form on the bit; place in the furnace immediately. The white residue should melt the instant the bit is inserted into the fire again; if it does not melt at once, reheat and repeat the operation.

The Sentinel paste is a trade composition consisting of molecular mixtures of metallic salts which melt at predetermined temperatures. The finely ground salts are mixed with paraffin wax. Before using it is heated gently so as to just melt the wax and reduce it to the consistency of thick paint, so it can then be streaked or painted on the work in the cold, or, as in the present case, the work is covered with a thin film of paste by dipping in the paste while the work is at a temperature above the melting point of the salts. When the wax carrying the Sentinel powder burns off, it leaves behind a layer of white salt, and upon the desired temperature being attained, the salt fuses and disappears, or, on a reduction of temperature the paste reappears as a white coating.

After the bit is put in the furnace and the white residue melts instantly, plunge the bit into cold water to about  $\frac{1}{4}$  to  $\frac{1}{2}$  in.

All bits in these tests were plunged into water as described above and slowly moved about in the water until fairly cool; then the steel was left in water until cold.

#### EXPERIMENTS WITH DOWN HOLES

The first series of tests was made in down holes. The purpose of these tests was to ascertain the effect of gage or diameter of bit on cutting speed. It was necessary to find this relation in order to compare different bits by reducing them to the same gage.

All tests were made in shallow holes, about 1 ft. in depth, so that the cuttings could be easily blown from the hole.

Down-hole tests were made only with the 4-point bit and at a pressure of 95 lb. In most cases, five 1-min. runs were made with a bit.

About 30 tests were made altogether. Table 1 shows the results of 12 of the most consistent of these tests.

TABLE 1.—*Results from Down Holes, 95 Lb. Pressure*

| No. of Test | Gage at Start | Meas. of Hole | Dist. Per Min. | Gage at Start | Meas. of Hole | Dist. Per Min. | Gage at Start | Meas. of Hole | Dist. Per Min. |
|-------------|---------------|---------------|----------------|---------------|---------------|----------------|---------------|---------------|----------------|
|             | 1.600         | 1.80          | .....          | 1.750         | 4.90          | .....          | 1.765         | 1.35          |                |
| 1           | 1.560         | 5.60          | 3.80           | 1.650         | 7.60          | 2.70           | 1.750         | 4.45          | 3.10           |
| 2           | 1.550         | 9.00          | 3.40           | 1.625         | 9.90          | 2.30           | 1.713         | 7.00          | 2.55           |
| 3           | 1.530         | 12.20         | 3.20           | 1.625         | 12.05         | 2.15           | 1.710         | 9.50          | 2.50           |
| 4           | 1.530         | 14.20         | 2.00           | 1.625         | 13.90         | 1.85           | 1.690         | 11.50         | 2.00           |
| 5           | 1.500         | 16.50         | 2.30           | 1.625         | 15.40         | 1.50           | 1.690         | 13.45         | 1.95           |
| 6           |               |               |                |               |               |                | 1.690         | 15.25         | 1.80           |
|             | 1.920         | 1.05          | .....          | 2.125         | 1.35          | .....          | 1.580         | 3.70          |                |
| 1           | 1.875         | 3.70          | 2.65           | 2.125         | 3.45          | 2.10           | 1.560         | 7.55          | 3.85           |
| 2           | 1.845         | 6.00          | 2.30           | 2.095         | 5.30          | 1.85           | 1.530         | 10.75         | 3.20           |
| 3           | 1.815         | 7.85          | 1.85           | 2.060         | 7.15          | 1.85           | 1.530         | 13.85         | 3.10           |
| 4           | 1.815         | 9.60          | 1.75           | 2.060         | 8.80          | 1.65           | 1.520         | 16.90         | 3.05           |
| 5           | 1.815         | 11.35         | 1.75           | 2.060         | 10.35         | 1.55           | 1.500         | 19.75         | 2.85           |
| 6           | 1.800         | 12.85         | 1.50           | 2.047         | 11.70         | 1.35           | 1.500         | 22.15         | 2.40           |
|             | 1.350         | 3.15          | .....          | 1.330         | 1.60          | .....          | 1.440         | 1.85          |                |
| 1           | 1.300         | 8.75          | 5.60           | 1.315         | 6.50          | 4.90           | 1.420         | 6.35          | 4.50           |
| 2           | 1.300         | 13.15         | 4.40           | 1.305         | 10.60         | 4.10           | 1.400         | 10.10         | 3.75           |
| 3           | 1.300         | 17.45         | 4.30           | 1.300         | 14.80         | 4.20           | 1.370         | 13.10         | 3.00           |
| 4           | 1.300         | 21.35         | 3.90           | 1.245         | 18.10         | 3.30           | 1.370         | 16.15         | 3.05           |
| 5           | 1.240         | 24.90         | 3.55           | 1.240         | 21.25         | 3.15           | 1.370         | 18.95         | 2.80           |
| 6           | 1.230         | 28.30         | 3.40           | 1.235         | 23.80         | 2.55           | 1.360         | 21.40         | 2.45           |
| 7           | 1.225         | 31.60         | 3.30           | 1.230         | 26.90         | 3.10           | 1.350         | 23.75         | 2.25           |
|             | New hole      |               |                |               |               |                |               |               |                |
| 8           | 1.220         | 6.95          | 0.00           | 1.230         | 29.70         | 2.80           | 1.320         | 25.95         | 2.20           |
| 9           | 1.215         | 10.85         | 3.90           |               |               |                |               |               |                |
|             | 1.470         | 1.85          | .....          | 1.590         | 2.10          | .....          | 1.560         | 6.10          |                |
| 1           | 1.440         | 5.85          | 4.00           | 1.560         | 5.80          | 3.70           | 1.550         | 9.80          | 3.70           |
| 2           | 1.440         | 9.15          | 3.30           | 1.520         | 9.05          | 3.25           | 1.530         | 12.70         | 2.90           |
| 3           | 1.400         | 12.00         | 2.85           | 1.500         | 12.05         | 3.00           | 1.500         | 15.20         | 2.50           |
| 4           | 1.370         | 14.55         | 2.55           | 1.490         | 14.90         | 2.85           | 1.500         | 17.45         | 2.25           |
| 5           | 1.370         | 17.00         | 2.45           | 1.490         | 17.50         | 2.60           | 1.500         | 19.70         | 2.25           |

The accompanying curve (Fig. 3) was plotted assuming that the cutting speed varies inversely as the square of the diameter. The first point on the theoretic curve was taken at gage 1.3 and speed 4.3, and the curve plotted accordingly. The results of the second test were then plotted in the same scale, the points falling close to the curve as shown. The second test was taken as being the most representative. Points plotted from the third and fourth tests fell close to a similar curve. In order to compare cutting speed of different gages, every condition, including depth of hole, etc., must be uniform, and it would not be fair to compare the speed of

a new bit with the speed of the same bit when run to a smaller gage. From this curve it is evident that for small gages the cutting speed varies inversely as the square of the diameter of the bit. For large gages the results do not seem to check so well, and from results obtained in former work by H. Vogel, the relation of the  $\frac{3}{2}$  power was obtained, which may apply to larger gages.

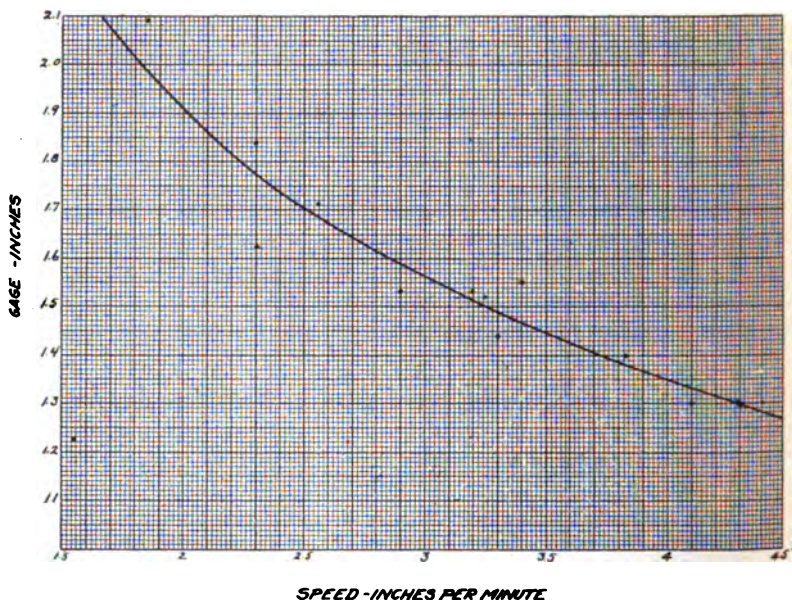


FIG. 3.—THEORETIC CURVE ASSUMING SPEED VARIES INVERSELY AS SQUARE OF DIAMETER, AND ACTUAL RESULTS FROM DIFFERENT GAGES IN DOWN HOLES.

#### RESULTS FROM UP HOLES

Three different pressures were used in drilling the up holes, 95 lb., 85 lb. and 70 lb. Most of the tests were made at the 95-lb. pressure. Nearly 200 different tests were made in all, but the tabulated results show only those that were most consistent.

All holes were first collared to about  $\frac{1}{2}$  in. depth; the bit to be tested was then run at a low pressure for about 6 sec. The bit was then taken out, the gage and the depth of hole measured, and the test was ready to start. One-minute runs by stop watch were made. All bits were run for four 1-min. tests and in a few cases six or eight 1-min. runs were made. No attempt was made to run each bit to its full extent, although in most cases at the end of four runs the cutting speed had materially diminished. The bits, however, probably would have drilled several inches farther, and in the case of the Carr bit, considerably farther. The life of bits seems to be the same at all pressures. That is, they will drill a certain distance and no more, no matter what the pressure is.



## RESULTS AT 95 LB. PRESSURE

Table 2 shows the actual results as obtained on the four different bits at 95 lb. pressure.

TABLE 2.—*Results at 95 Lb. Pressure*

## 4-POINT STARTER

| Gage at Start,<br>Inches | Meas. of<br>Hole,<br>Inches | Dist. Drilled<br>Per Min.,<br>Inches | Average<br>Diam. | Total Dist.<br>Drilled | Average<br>Speed<br>Per Min. | Loss in<br>Gage<br>Per Inch. |
|--------------------------|-----------------------------|--------------------------------------|------------------|------------------------|------------------------------|------------------------------|
| 1.984                    | 11.55                       |                                      |                  |                        |                              |                              |
| 1.920                    | 14.00                       | 2.45                                 |                  |                        |                              |                              |
| 1.908                    | 16.10                       | 2.10                                 |                  |                        |                              |                              |
| 1.908                    | 18.15                       | 2.05                                 |                  |                        |                              |                              |
| 1.908                    | 20.10                       | 1.95                                 | 1.9230           | 8.55                   | 2.1370                       | 0.0089                       |
| 1.921                    | 13.40                       |                                      |                  |                        |                              |                              |
| 1.875                    | 16.00                       | 2.60                                 |                  |                        |                              |                              |
| 1.875                    | 18.30                       | 2.30                                 |                  |                        |                              |                              |
| 1.844                    | 20.40                       | 2.10                                 |                  |                        |                              |                              |
| 1.830                    | 22.20                       | 1.80                                 | 1.8700           | 8.80                   | 2.2000                       | 0.0102                       |
| 1.938                    | 11.60                       |                                      |                  |                        |                              |                              |
| 1.858                    | 14.05                       | 2.45                                 |                  |                        |                              |                              |
| 1.844                    | 16.30                       | 2.35                                 |                  |                        |                              |                              |
| 1.812                    | 18.60                       | 2.30                                 |                  |                        |                              |                              |
| 1.812                    | 20.85                       | 2.25                                 | 1.8480           | 9.35                   | 2.3370                       | 0.0124                       |
| 1.938                    | 12.40                       |                                      |                  |                        |                              |                              |
| 1.908                    | 14.90                       | 2.50                                 |                  |                        |                              |                              |
| 1.875                    | 17.20                       | 2.30                                 |                  |                        |                              |                              |
| 1.844                    | 19.25                       | 2.05                                 |                  |                        |                              |                              |
| 1.830                    | 21.10                       | 1.85                                 | 1.8790           | 8.70                   | 2.1750                       | 0.0124                       |

## 4-POINT SECOND

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.750 | 18.60 |      |        |       |        |        |
| 1.720 | 22.45 | 3.85 |        |       |        |        |
| 1.625 | 25.60 | 3.15 |        |       |        |        |
| 1.610 | 28.15 | 2.55 |        |       |        |        |
| 1.594 | 30.15 | 2.10 | 1.6702 | 11.65 | 2.9120 | 0.0133 |
| 1.688 | 18.80 |      |        |       |        |        |
| 1.625 | 22.25 | 3.45 |        |       |        |        |
| 1.625 | 25.25 | 3.00 |        |       |        |        |
| 1.610 | 27.65 | 2.40 |        |       |        |        |
| 1.594 | 29.60 | 1.95 | 1.6290 | 10.80 | 2.7000 | 0.0087 |
| 1.750 | 19.55 |      |        |       |        |        |
| 1.720 | 23.30 | 3.75 |        |       |        |        |
| 1.658 | 26.65 | 3.35 |        |       |        |        |
| 1.625 | 29.35 | 2.70 |        |       |        |        |
| 1.625 | 31.60 | 2.25 | 1.6560 | 12.05 | 2.9820 | 0.0104 |

TABLE 2.—Results at 95 Lb. Pressure.—(Continued)

## 4-POINT THIRD

| Gage at Start,<br>Inches | Meas. of<br>Hole,<br>Inches | Dist. Drilled<br>Per<br>Min.,<br>Inches | Average<br>Diam. | Total Dist.<br>Drilled | Average<br>Speed<br>Per Min. | Loss in<br>Gage<br>Per Inch |
|--------------------------|-----------------------------|---|------------------|------------------------|------------------------------|-----------------------------|
| 1.437                    | 31.35                       |   |                  |                        |                              |                             |
| 1.375                    | 35.95                       | 4.60                                    |                  |                        |                              |                             |
| 1.375                    | 40.00                       | 4.05                                    |                  |                        |                              |                             |
| 1.359                    | 43.40                       | 3.40                                    |                  |                        |                              |                             |
| 1.328                    | 46.60                       | 3.20                                    | 1.3732           | 15.25                  | 3.8125                       | 0.0071                      |
| 1.453                    | 32.30                       |   |                  |                        |                              |                             |
| 1.422                    | 37.20                       | 4.90                                    |                  |                        |                              |                             |
| 1.390                    | 41.35                       | 4.15                                    |                  |                        |                              |                             |
| 1.359                    | 44.95                       | 3.30                                    |                  |                        |                              |                             |
| 1.359                    | 47.90                       | 2.95                                    | 1.3999           | 15.30                  | 3.8250                       | 0.0061                      |
| 1.437                    | 32.10                       |   |                  |                        |                              |                             |
| 1.375                    | 37.05                       | 4.95                                    |                  |                        |                              |                             |
| 1.344                    | 41.30                       | 4.25                                    |                  |                        |                              |                             |
| 1.312                    | 44.80                       | 3.50                                    |                  |                        |                              |                             |
| 1.312                    | 47.90                       | 3.10                                    | 1.3576           | 15.80                  | 3.9500                       | 0.0078                      |
| 1.469                    | 30.05                       |   |                  |                        |                              |                             |
| 1.437                    | 34.80                       | 4.75                                    |                  |                        |                              |                             |
| 1.406                    | 38.90                       | 4.10                                    |                  |                        |                              |                             |
| 1.390                    | 42.35                       | 3.45                                    |                  |                        |                              |                             |
| 1.375                    | 45.35                       | 3.00                                    | 1.4117           | 15.30                  | 3.8250                       | 0.0061                      |

## 6-POINT STARTER

|       |       |      |        |      |        |        |
|-------|-------|------|--------|------|--------|--------|
| 1.938 | 11.60 |      |        |      |        |        |
| 1.894 | 13.85 | 2.25 |        |      |        |        |
| 1.858 | 15.60 | 1.95 |        |      |        |        |
| 1.844 | 17.55 | 1.75 |        |      |        |        |
| 1.844 | 19.05 | 1.50 | 1.8660 | 7.45 | 1.8620 | 0.0126 |
| 1.970 | 12.20 |      |        |      |        |        |
| 1.920 | 14.35 | 2.15 |        |      |        |        |
| 1.875 | 16.20 | 1.85 |        |      |        |        |
| 1.830 | 17.95 | 1.75 |        |      |        |        |
| 1.830 | 19.55 | 1.60 | 1.8380 | 7.35 | 1.8830 | 0.0190 |
| 1.938 | 12.80 |      |        |      |        |        |
| 1.894 | 15.05 | 2.25 |        |      |        |        |
| 1.858 | 16.80 | 1.75 |        |      |        |        |
| 1.844 | 18.40 | 1.60 |        |      |        |        |
| 1.830 | 19.85 | 1.45 | 1.8740 | 7.05 | 1.7620 | 0.0153 |

TABLE 2.—*Results at 95 Lb. Pressure.—(Continued)*

## 6-POINT STARTER.

| Gage at Start,<br>Inches | Meas. of<br>Hole,<br>Inches | Dist. Drilled<br>Per Min.,<br>Inches | Average<br>Diam. | Total Dist.<br>Drilled | Average<br>Speed<br>Per Min. | Loss in<br>Gage<br>Per Inch |
|--------------------------|-----------------------------|--------------------------------------|------------------|------------------------|------------------------------|-----------------------------|
| 2.002                    | 12.15                       |                                      |                  |                        |                              |                             |
| 1.938                    | 14.30                       | 2.15                                 |                  |                        |                              |                             |
| 1.908                    | 16.30                       | 2.00                                 |                  |                        |                              |                             |
| 1.875                    | 18.15                       | 1.85                                 |                  |                        |                              |                             |
| 1.858                    | 19.75                       | 1.60                                 | 1.8970           | 7.60                   | 1.9000                       | 0.0289                      |

## 6-POINT SECOND

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.720 | 18.60 |      |        |       |        |        |
| 1.688 | 21.55 | 2.95 |        |       |        |        |
| 1.641 | 24.25 | 2.70 |        |       |        |        |
| 1.625 | 26.80 | 2.55 |        |       |        |        |
| 1.625 | 29.00 | 2.25 | 1.6680 | 10.45 | 2.6120 | 0.0091 |
| 1.720 | 18.55 |      |        |       |        |        |
| 1.672 | 21.25 | 2.70 |        |       |        |        |
| 1.641 | 23.80 | 2.55 |        |       |        |        |
| 1.625 | 26.15 | 2.35 |        |       |        |        |
| 1.625 | 28.30 | 2.15 | 1.6550 | 9.75  | 2.4370 | 0.0097 |
| 1.750 | 19.50 |      |        |       |        |        |
| 1.704 | 22.40 | 2.90 |        |       |        |        |
| 1.658 | 25.15 | 2.75 |        |       |        |        |
| 1.625 | 27.75 | 2.60 |        |       |        |        |
| 1.610 | 30.10 | 2.35 | 1.6800 | 10.60 | 2.6500 | 0.0132 |
| 1.735 | 20.00 |      |        |       |        |        |
| 1.672 | 22.70 | 2.70 |        |       |        |        |
| 1.641 | 25.25 | 2.55 |        |       |        |        |
| 1.625 | 27.55 | 2.30 |        |       |        |        |
| 1.625 | 29.50 | 1.95 | 1.6440 | 9.50  | 2.3750 | 0.0116 |

## 6-POINT THIRD

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.485 | 39.55 |      |        |       |        |        |
| 1.453 | 42.85 | 3.30 |        |       |        |        |
| 1.437 | 46.00 | 3.15 |        |       |        |        |
| 1.422 | 48.85 | 2.85 |        |       |        |        |
| 1.422 | 51.15 | 2.30 | 1.4444 | 11.60 | 2.9000 | 0.0054 |
| 1.437 | 28.55 |      |        |       |        |        |
| 1.375 | 31.75 | 3.20 |        |       |        |        |
| 1.375 | 34.80 | 3.05 |        |       |        |        |
| 1.375 | 37.75 | 2.85 |        |       |        |        |
| 1.344 | 40.40 | 2.65 | 1.3713 | 11.75 | 2.9375 | 0.0079 |

TABLE 2.—*Results at 95 Lb. Pressure.—(Continued)*

## 6-POINT THIRD.

| Gage at Start,<br>Inches | Meas. of<br>Hole,<br>Inches | Dist. Drilled<br>Per Min.,<br>Inches | Average<br>Diam. | Total Dist.<br>Drilled | Average<br>Speed<br>Per Min. | Loss in<br>Gage<br>Per Inch |
|--------------------------|-----------------------------|--------------------------------------|------------------|------------------------|------------------------------|-----------------------------|
| 1.469                    | 33.35                       |                                      |                  |                        |                              |                             |
| 1.437                    | 36.45                       | 3.10                                 |                  |                        |                              |                             |
| 1.422                    | 39.30                       | 2.85                                 |                  |                        |                              |                             |
| 1.422                    | 41.30                       | 2.00                                 |                  |                        |                              |                             |
| 1.406                    | 43.10                       | 1.80                                 | 1.4327           | 9.75                   | 2.4375                       | 0.0064                      |
| 1.500                    | 32.10                       |                                      |                  |                        |                              |                             |
| 1.469                    | 34.95                       | 2.85                                 |                  |                        |                              |                             |
| 1.453                    | 37.75                       | 2.80                                 |                  |                        |                              |                             |
| 1.437                    | 40.35                       | 2.60                                 |                  |                        |                              |                             |
| 1.422                    | 42.40                       | 2.05                                 | 1.4571           | 10.30                  | 2.5783                       | 0.0075                      |

## CARR BIT

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.547 | 20.80 |      |        |       |        |        |
| 1.547 | 25.00 | 4.20 |        |       |        |        |
| 1.547 | 28.30 | 3.30 |        |       |        |        |
| 1.532 | 31.20 | 2.90 |        |       |        |        |
| 1.516 | 33.75 | 2.55 |        |       |        |        |
| 1.510 | 36.30 | 2.55 |        |       |        |        |
| 1.500 | 38.80 | 2.50 |        |       |        |        |
| 1.490 | 41.20 | 2.40 | 1.5408 | 12.95 | 3.2400 | 0.0028 |
| 1.547 | 28.45 |      |        |       |        |        |
| 1.547 | 32.30 | 3.85 |        |       |        |        |
| 1.532 | 34.80 | 2.50 |        |       |        |        |
| 1.510 | 37.45 | 2.55 |        |       |        |        |
| 1.510 | 40.45 | 3.00 |        |       |        |        |
| 1.510 | 43.00 | 2.55 |        |       |        |        |
| 1.485 | 45.50 | 2.50 |        |       |        |        |
| 1.485 | 48.00 | 2.50 | 1.5300 | 11.90 | 2.9770 | 0.0032 |
| 1.624 | 12.55 |      |        |       |        |        |
| 1.609 | 16.15 | 3.60 |        |       |        |        |
| 1.594 | 20.00 | 3.85 |        |       |        |        |
| 1.594 | 22.95 | 2.95 |        |       |        |        |
| 1.578 | 25.70 | 2.75 | 1.6010 | 13.15 | 3.2900 | 0.0042 |
| 1.623 | 13.10 |      |        |       |        |        |
| 1.609 | 16.50 | 3.40 |        |       |        |        |
| 1.609 | 20.20 | 3.70 |        |       |        |        |
| 1.609 | 23.70 | 3.50 |        |       |        |        |
| 1.594 | 26.70 | 3.00 | 1.6090 | 13.60 | 3.4000 | 0.0021 |

TABLE 2.—*Results at 95 Lb. Pressure.—(Continued)*

## Z BIT

| Gage at Start,<br>Inches | Meas. of<br>Hole,<br>Inches | Dist. Drilled<br>Per Min.,<br>Inches | Average<br>Diam. | Total Dist.<br>Drilled | Average<br>Speed<br>Per Min. | Loss in<br>Gage<br>Per Inch |
|--------------------------|-----------------------------|--------------------------------------|------------------|------------------------|------------------------------|-----------------------------|
| 1.688                    | 14.10                       |                                      |                  |                        |                              |                             |
| 1.641                    | 17.25                       | 3.15                                 |                  |                        |                              |                             |
| 1.610                    | 19.65                       | 2.40                                 |                  |                        |                              |                             |
| 1.594                    | 21.80                       | 2.15                                 |                  |                        |                              |                             |
| 1.594                    | 23.70                       | 1.90                                 | 1.6160           | 9.60                   | 2.3920                       | 0.0098                      |
| 1.672                    | 13.50                       |                                      |                  |                        |                              |                             |
| 1.625                    | 16.90                       | 3.40                                 |                  |                        |                              |                             |
| 1.594                    | 19.20                       | 2.30                                 |                  |                        |                              |                             |
| 1.562                    | 21.40                       | 2.20                                 |                  |                        |                              |                             |
| 1.562                    | 23.35                       | 1.95                                 | 1.6010           | 9.85                   | 2.4650                       | 0.0111                      |
| 1.501                    | 13.60                       |                                      |                  |                        |                              |                             |
| 1.485                    | 17.60                       | 4.00                                 |                  |                        |                              |                             |
| 1.485                    | 20.95                       | 3.35                                 |                  |                        |                              |                             |
| 1.469                    | 23.95                       | 3.00                                 |                  |                        |                              |                             |
| 1.469                    | 26.50                       | 2.55                                 | 1.4800           | 12.90                  | 3.2280                       | 0.0025                      |
| 1.469                    | 14.10                       |                                      |                  |                        |                              |                             |
| 1.469                    | 17.80                       | 3.70                                 |                  |                        |                              |                             |
| 1.375                    | 21.05                       | 3.15                                 |                  |                        |                              |                             |
| 1.375                    | 24.30                       | 3.25                                 |                  |                        |                              |                             |
| 1.375                    | 26.95                       | 2.65                                 | 1.4130           | 12.75                  | 3.1960                       | 0.0073                      |

Table 3 represents the summary of averages shown in Table 2, and also the same averages reduced, for the purpose of comparison, to uniform diameters by assuming that the cutting speed varies inversely as the square of the diameter.

TABLE 3.—*Summary of Results at 95 Lb. Pressure*Reduced to uniform gages, assuming speed varies as  $\frac{1}{D^2}$ 

| Kind of Bit                 | Average<br>Diam.,<br>Inches | Average<br>Dist.<br>Drilled | Average<br>Speed<br>Per Min. | Average<br>Initial<br>Speed | Average<br>Final<br>Speed | Average Loss<br>in Gage<br>Per Inch |
|-----------------------------|-----------------------------|-----------------------------|------------------------------|-----------------------------|---------------------------|-------------------------------------|
| A 4-point starter.....      | 1.880                       | 8.85                        | 2.212                        | 2.500                       | 1.962                     | 0.0110                              |
| B 4-point second.....       | 1.652                       | 11.50                       | 2.865                        | 3.683                       | 2.100                     | 0.0108                              |
| C 4-point third.....        | 1.385                       | 15.41                       | 3.853                        | 4.800                       | 3.062                     | 0.0068                              |
| Reduced to uniform diameter |                             |                             |                              |                             |                           |                                     |
| From A.....                 | 1.750                       | .....                       | 2.554                        |                             |                           |                                     |
| From B.....                 | 1.750                       | .....                       | 2.552                        |                             |                           |                                     |
| From C.....                 | 1.750                       | .....                       | 2.415                        |                             |                           |                                     |
|                             | 1.750                       | .....                       | 2.507                        | Average of 4-point          |                           |                                     |
| A 6-point starter.....      | 1.869                       | 7.36                        | 1.852                        | 2.200                       | 1.537                     | 0.0189                              |
| B 6-point second.....       | 1.662                       | 10.07                       | 2.519                        | 2.812                       | 2.175                     | 0.0109                              |
| C 6-point third.....        | 1.426                       | 10.85                       | 2.713                        | 3.112                       | 2.200                     | 0.0068                              |
| Reduced to uniform diameter |                             |                             |                              |                             |                           |                                     |
| From A.....                 | 1.750                       | .....                       | 2.112                        |                             |                           |                                     |
| From B.....                 | 1.750                       | .....                       | 2.271                        |                             |                           |                                     |
| From C.....                 | 1.750                       | .....                       | 1.802                        |                             |                           |                                     |
|                             | 1.750                       | .....                       | 2.062                        | Average of 6-point          |                           |                                     |
| A Carr bit second.....      | 1.535                       | 12.42                       | 3.109                        | 4.025                       | 2.775                     | 0.0028                              |
| B Carr bit third.....       | 1.605                       | 13.37                       | 3.345                        | 3.500                       | 2.875                     | 0.0032                              |
| Reduced to uniform diameter |                             |                             |                              |                             |                           |                                     |
| From A.....                 | 1.750                       | .....                       | 2.394                        |                             |                           |                                     |
| From B.....                 | 1.750                       | .....                       | 2.488                        |                             |                           |                                     |
|                             | 1.750                       | .....                       | 2.441                        | Average Carr bit            |                           |                                     |
| A Z bit second.....         | 1.609                       | 9.72                        | 2.429                        | 3.275                       | 1.925                     | 0.0104                              |
| B Z bit third.....          | 1.447                       | 12.87                       | 3.212                        | 3.850                       | 2.600                     | 0.0049                              |
| Reduced to uniform diameter |                             |                             |                              |                             |                           |                                     |
| From A.....                 | 1.750                       | .....                       | 2.053                        |                             |                           |                                     |
| From B.....                 | 1.750                       | .....                       | 2.196                        |                             |                           |                                     |
|                             | 1.750                       | .....                       | 2.130                        | Average Z bit               |                           |                                     |

It will be seen from these results that the comparative cutting speeds of the different bits at this pressure are as follows:

- (1) 4-point (gage 1.75)..... 2.51 in. per minute
- (2) 6-point (gage 1.75)..... 2.06 in. per minute
- (2) Carr (gage 1.75)..... 2.44 in. per minute
- (3) Z (gage 1.75)..... 2.13 in. per minute

At this pressure the 4-point bit cuts slightly faster than the Carr. It was very difficult to make the Z bit hold up at all on the high pressure, as the corners would invariably chip off.

## RESULTS AT 85 LB. AND 70 LB. PRESSURE

Table 4 shows the results obtained using 85 lb. pressure and Table 5 the results at 70 lb.

TABLE 4.—*Results at 85 Lb. Pressure*

## 6-POINT BIT

| Gage at Start,<br>Inches | Meas. of<br>Hole,<br>Inches | Dist. Drilled<br>Per Min.,<br>Inches | Average<br>Diam. | Total Dist.<br>Drilled | Average<br>Speed<br>Per Min. | Loss in<br>Gage<br>Per Inch |
|--------------------------|-----------------------------|--------------------------------------|------------------|------------------------|------------------------------|-----------------------------|
| 1.485                    | 3.10                        |                                      |                  |                        |                              |                             |
| 1.453                    | 6.10                        | 3.00                                 |                  |                        |                              |                             |
| 1.437                    | 8.45                        | 2.35                                 |                  |                        |                              |                             |
| 1.422                    | 10.70                       | 2.25                                 |                  |                        |                              |                             |
| 1.422                    | 12.55                       | 1.85                                 | 1.4444           | 9.45                   | 2.3670                       | 0.0066                      |

## 4-POINT BIT

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.437 | 14.75 |      |        |       |        |        |
| 1.422 | 18.60 | 3.85 |        |       |        |        |
| 1.390 | 21.65 | 3.05 |        |       |        |        |
| 1.375 | 24.00 | 2.35 |        |       |        |        |
| 1.359 | 26.00 | 2.00 | 1.4020 | 11.25 | 2.8240 | 0.0069 |

## Z BIT

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.485 | 13.70 |      |        |       |        |        |
| 1.437 | 16.90 | 3.20 |        |       |        |        |
| 1.437 | 19.90 | 2.00 |        |       |        |        |
| 1.437 | 22.80 | 2.90 |        |       |        |        |
| 1.422 | 25.35 | 2.55 | 1.4420 | 11.65 | 2.9140 | 0.0054 |

## 4-POINT 4° TAPER

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.422 | 39.00 |      |        |       |        |        |
| 1.406 | 42.05 | 3.35 |        |       |        |        |
| 1.390 | 25.25 | 2.30 |        |       |        |        |
| 1.375 | 47.90 | 2.65 |        |       |        |        |
| 1.375 | 50.10 | 2.20 | 1.3950 | 11.40 | 2.8546 | 0.0041 |

## Z BIT FILED

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.390 | 14.15 |      |        |       |        |        |
| 1.359 | 18.20 | 4.05 |        |       |        |        |
| 1.359 | 21.95 | 3.75 |        |       |        |        |
| 1.344 | 24.45 | 3.50 |        |       |        |        |
| 1.344 | 26.90 | 2.40 | 1.3580 | 13.70 | 3.3458 | 0.0033 |

## CARR BIT

|       |       |      |        |       |        |        |
|-------|-------|------|--------|-------|--------|--------|
| 1.453 | 14.15 |      |        |       |        |        |
| 1.437 | 17.65 | 3.50 |        |       |        |        |
| 1.422 | 21.05 | 3.40 |        |       |        |        |
| 1.422 | 23.95 | 2.90 |        |       |        |        |
| 1.422 | 26.25 | 2.30 | 1.4310 | 12.10 | 3.0275 | 0.0026 |

TABLE 5.—*Results at 70 Lb. Pressure*

| 4-POINT BIT              |                             |                                      |                  |                        |                              |                          |
|--------------------------|-----------------------------|--------------------------------------|------------------|------------------------|------------------------------|--------------------------|
| Gage at Start,<br>Inches | Meas. of<br>Hole,<br>Inches | Dist. Drilled<br>Per Min.,<br>Inches | Average<br>Diam. | Total Dist.<br>Drilled | Average<br>Speed<br>Per Min. | Loss in Gage<br>Per Inch |
| 1.62                     | 13.00                       |                                      |                  |                        |                              |                          |
| 1.61                     | 15.45                       | 2.15                                 |                  |                        |                              |                          |
| 1.59                     | 17.15                       | 1.70                                 |                  |                        |                              |                          |
| 1.56                     | 18.65                       | 1.50                                 |                  |                        |                              |                          |
| 1.56                     | 20.05                       | 1.40                                 | 1.5930           | 6.75                   | 1.6910                       | 0.0093                   |
| 6-POINT BIT              |                             |                                      |                  |                        |                              |                          |
| 1.50                     | 13.60                       |                                      |                  |                        |                              |                          |
| 1.48                     | 15.15                       | 1.55                                 |                  |                        |                              |                          |
| 1.47                     | 16.40                       | 1.25                                 |                  |                        |                              |                          |
| 1.45                     | 17.55                       | 1.15                                 |                  |                        |                              |                          |
| 1.44                     | 18.60                       | 1.05                                 | 1.4710           | 5.00                   | 1.2520                       | 0.0126                   |
| CARR BIT                 |                             |                                      |                  |                        |                              |                          |
| 1.44                     | 13.60                       |                                      |                  |                        |                              |                          |
| 1.44                     | 16.40                       | 2.80                                 |                  |                        |                              |                          |
| 1.42                     | 18.60                       | 2.20                                 |                  |                        |                              |                          |
| 1.42                     | 20.60                       | 2.00                                 |                  |                        |                              |                          |
| 1.42                     | 22.60                       | 2.00                                 | 1.4320           | 9.00                   | 2.2570                       | 0.0016                   |
| Z BIT                    |                             |                                      |                  |                        |                              |                          |
| 1.477                    | 17.20                       |                                      |                  |                        |                              |                          |
| 1.477                    | 20.15                       | 2.95                                 |                  |                        |                              |                          |
| 1.422                    | 22.70                       | 2.20                                 |                  |                        |                              |                          |
| 1.422                    | 25.10                       | 2.40                                 |                  |                        |                              |                          |
| 1.406                    | 26.65                       | 1.55                                 | 1.4270           | 9.10                   | 2.2780                       | 0.0034                   |

Table 6 shows the final average reduced to three different gages by the square of diameter.



TABLE 6.—*Comparative Cutting Speeds of Different Bits*Reduced to uniform diameters, assuming speed varies  $\frac{1}{D^2}$ 

| Kind of Bit           | Reduced Gage, Inches | Average Speed Per Min., 95 Lb. P. | Average Speed Per Min., 85 Lb. P. | Average Speed Per Min., 70 Lb. P. |
|-----------------------|----------------------|-----------------------------------|-----------------------------------|-----------------------------------|
| 4-point.....          | 2.00                 | 1.92                              | 1.39                              | 0.85                              |
| 6-point.....          | 2.00                 | 1.59                              | 1.23                              | 0.68                              |
| Carr.....             | 2.00                 | 1.87                              | 1.56                              | 1.16                              |
| Z.....                | 2.00                 | 1.63                              | 1.51                              | 1.16                              |
| Z flat angle.....     | 2.00                 | .....                             | 1.54                              | .....                             |
| 4-point 5° taper..... | 2.00                 | .....                             | 1.39                              | .....                             |
| 4-point.....          | 1.75                 | 2.51                              | 1.81                              | 1.11                              |
| 6-point.....          | 1.75                 | 2.06                              | 1.61                              | 0.88                              |
| Carr.....             | 1.75                 | 2.44                              | 2.03                              | 1.51                              |
| Z.....                | 1.75                 | 2.13                              | 1.97                              | 1.51                              |
| Z flat angle.....     | 1.75                 | .....                             | 2.01                              | .....                             |
| 4-point 5° taper..... | 1.75                 | .....                             | 1.81                              | .....                             |
| 4-point.....          | 1.50                 | 3.41                              | 2.47                              | 1.51                              |
| 6-point.....          | 1.50                 | 2.82                              | 2.19                              | 1.20                              |
| Carr.....             | 1.50                 | 3.32                              | 2.77                              | 2.06                              |
| Z.....                | 1.50                 | 2.90                              | 2.69                              | 2.06                              |
| Z flat angle.....     | 1.50                 | .....                             | 2.74                              | .....                             |
| 4-point 5° taper..... | 1.50                 | .....                             | 2.47                              | .....                             |

The curves in Figs. 4 and 5 show graphically the comparative cutting speeds of the different bits at different pressures. These were plotted by using the averages from bits of 1½-in. gage.

Figs. 6, 7 and 8 show the variation in cutting speed with gage for the different bits and at the three pressures.

### THEORY OF DRILLING ROCK

According to B. F. Tillson,<sup>1</sup> "when rock is excavated by a drill bit three applications of forces seem to be involved—by abrasion, by crushing, and by severing or chipping. Although all of these must take place to a certain degree, the greatest amount of useful work is performed when the percentage of force applied to chip reaches a maximum." According to this theory, the screen analysis of cutting ought to show the most efficient bit. Samples were taken from the up holes when drilling at 95 lb. pressure, from which the screen analyses shown in Table 7 were made. Screening was done in Tyler Ro-Tap apparatus. The samples were taken near the first part of a run with each bit. Some of the dust was lost in taking the sample, but as much as possible of the entire sample was saved.

<sup>1</sup> Testing and Application of Hammer Drills. *Trans.* (1915), 51, 251.

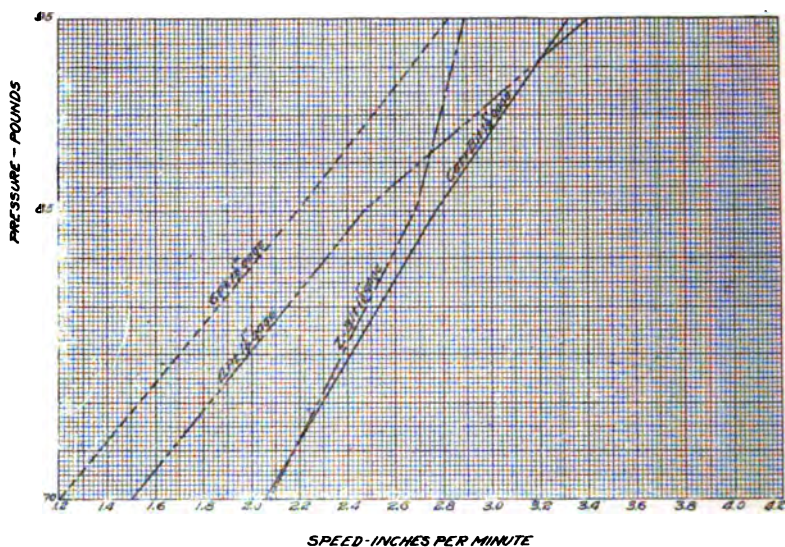


FIG. 4.—RELATIVE CUTTING SPEEDS OF 6-FT., 4-FT., CARR AND Z BITS AT DIFFERENT PRESSURES.

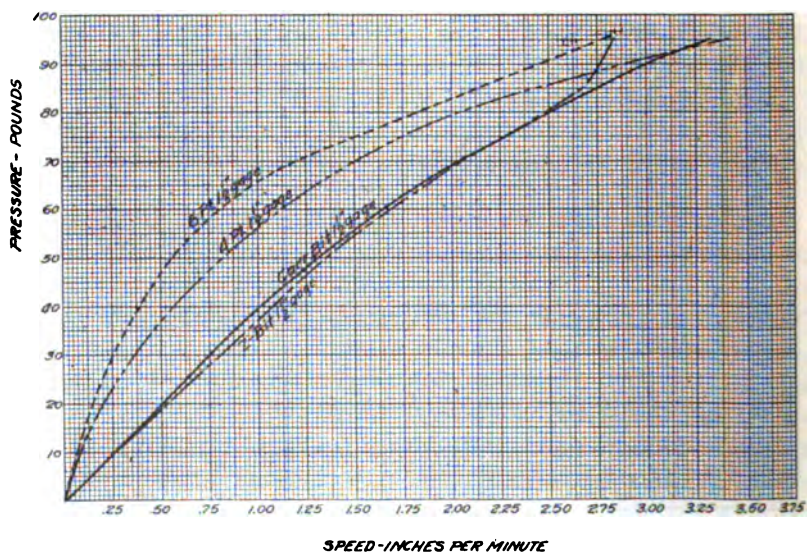


FIG. 5.—RELATIVE CUTTING SPEEDS OF 6-FT., 4-FT., CARR AND Z BITS AT PRESSURES FROM 10 TO 95 LB.

Table 7.—Screen Analysis of Cuttings From Up Holes, 95 Lb. Pressure

| On   | Opening | 6-Point Bit |                   | Z Bit (Filed) |                   | Z Bit<br>1¼ In. Diameter |                   | 4-Point<br>5° Taper |                   |
|------|---------|-------------|-------------------|---------------|-------------------|--------------------------|-------------------|---------------------|-------------------|
|      |         | Per Cent.   | Per Cent.<br>Cum. | Per Cent.     | Per Cent.<br>Cum. | Per Cent.                | Per Cent.<br>Cum. | Per Cent.           | Per Cent.<br>Cum. |
| +6   | 0.1310  | .....       | .....             | 0.69          | 0.69              | 0.62                     | 0.62              |                     |                   |
| +8   | 0.0930  | .....       | .....             | 1.72          | 2.41              | 1.75                     | 2.37              | 1.09                | 1.09              |
| +10  | 0.0650  | 0.59        | 0.59              | 2.76          | 5.17              | 3.75                     | 6.12              | 0.73                | 1.82              |
| +14  | 0.0460  | 0.88        | 1.47              | 3.52          | 8.69              | 3.94                     | 10.06             | 1.86                | 3.68              |
| +20  | 0.0328  | 5.00        | 6.47              | 3.31          | 12.00             | 4.06                     | 14.12             | 2.52                | 6.20              |
| +28  | 0.0232  | 10.25       | 16.72             | 7.45          | 19.45             | 8.82                     | 22.94             | 4.67                | 10.87             |
| +35  | 0.0164  | 13.82       | 30.54             | 10.00         | 29.45             | 10.18                    | 33.12             | 6.68                | 17.55             |
| +48  | 0.0116  | 16.77       | 47.31             | 12.00         | 41.45             | 12.50                    | 45.62             | 9.71                | 27.26             |
| +65  | 0.0080  | 14.72       | 62.03             | 11.59         | 53.04             | 12.25                    | 57.87             | 14.75               | 42.01             |
| +100 | 0.0058  | 16.17       | 78.20             | 20.70         | 73.74             | 16.50                    | 74.37             | 22.30               | 64.31             |
| -100 | .....   | 21.75       | 99.95             | 26.10         | 99.84             | 25.60                    | 99.97             | 35.60               | 99.91             |

| On   | Opening | 4-Point Bit |                   | 6-Point Bit |                   | Carr Bit<br>1st Run |                   | Dull Carr Bit |                   |
|------|---------|-------------|-------------------|-------------|-------------------|---------------------|-------------------|---------------|-------------------|
|      |         | Per Cent.   | Per Cent.<br>Cum. | Per Cent.   | Per Cent.<br>Cum. | Per Cent.           | Per Cent.<br>Cum. | Per Cent.     | Per Cent.<br>Cum. |
| +6   | 0.1310  | 3.05        | 3.05              |             |                   |                     |                   |               |                   |
| +8   | 0.0930  | 1.61        | 4.66              | 1.15        | 1.15              | 7.10                | 7.10              | 5.45          | 5.45              |
| +10  | 0.0650  | 3.26        | 7.92              | 1.78        | 2.93              | 2.42                | 9.52              | 2.72          | 8.17              |
| +14  | 0.0460  | 3.09        | 11.01             | 1.00        | 3.93              | 3.31                | 12.83             | 3.38          | 11.55             |
| +20  | 0.0328  | 3.78        | 14.79             | 2.87        | 6.80              | 2.66                | 15.49             | 0.82          | 12.37             |
| +28  | 0.0232  | 8.48        | 23.27             | 6.90        | 13.70             | 5.45                | 20.94             | 4.74          | 17.11             |
| +35  | 0.0164  | 10.22       | 33.49             | 9.90        | 23.60             | 7.10                | 28.04             | 6.97          | 24.08             |
| +48  | 0.0116  | 12.55       | 46.04             | 13.80       | 37.40             | 8.20                | 36.24             | 6.43          | 30.51             |
| +65  | 0.0082  | 14.25       | 60.29             | 17.05       | 54.45             | 12.55               | 48.79             | 15.05         | 45.56             |
| +100 | 0.0058  | 15.75       | 76.04             | 21.05       | 75.50             | 15.55               | 64.34             | 15.92         | 61.48             |
| -100 | .....   | 23.90       | 99.94             | 24.40       | 99.89             | 35.75               | 100.09            | 38.60         | 100.08            |

The curves in Fig. 9 show graphically the results of screen analysis. It will be noted that the 6-point bit produces the finest cuttings, the Z bit the next finest, the Carr next and the 4-point the coarsest. By reference to Fig. 6, it will be seen that this corresponds to the relative cutting speeds, the 6-point slowest, the Z next, the Carr next and 4-point fastest.

In order to study further the cutting action of the different bits, four shallow holes about 1 in. deep were drilled beside each other under the same conditions and using 85 lb. pressure.

A study of the bottoms of the holes revealed the fact that the 6-point hole was perfectly smooth, the 4-point nearly as smooth, the Z bit and Carr bit holes were quite rough, the Carr bit having a conical shaped projection in the center due to the large hole in the bit. It will be noted

from Fig. 7 that the relative cutting speeds at 85 lb. pressure are as follows: 6-point, 4-point, Z and Carr, which corresponded with the apparent roughness of the bottoms of the holes.

The use of the large hole in the center of the Carr bit is evidently one of its advantages and the same idea could be applied to advantage on all other bits. The advantage of the large hole is that it leaves a center core and diminishes the cutting surface.

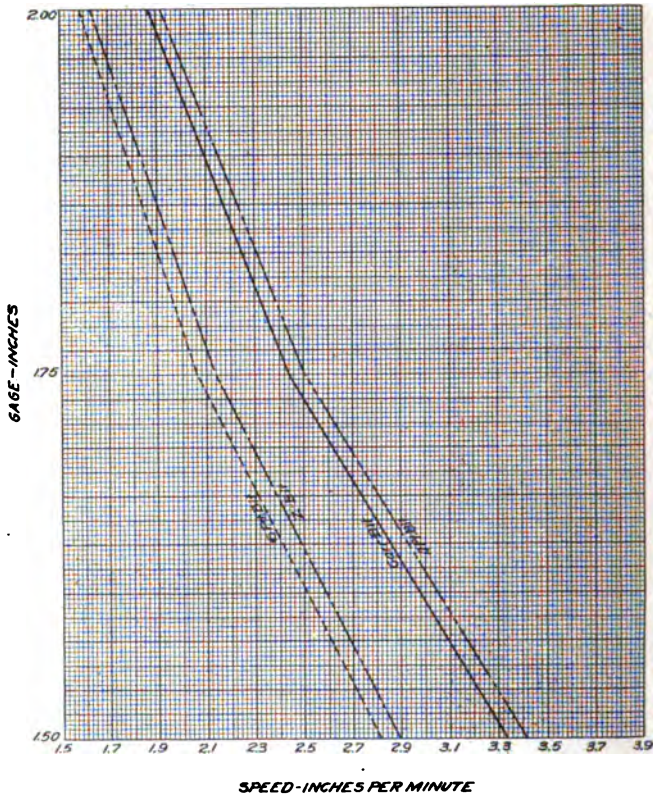


FIG. 6.—RELATIVE CUTTING SPEEDS OF 6-PT., Z, CARR AND 4-PT. BITS AT 95 LB. PRESSURE.

### CONCLUSIONS: COMPARATIVE MERITS OF EACH TYPE OF BIT

#### *Cutting Speeds*

|     | 95 Lb.  | 85 Lb.  | 70 Lb.  |
|-----|---------|---------|---------|
| (1) | 4-point | Carr    | Z bit.  |
| (2) | Carr    | Z       | Carr    |
| (3) | Z       | 4-point | 4-point |
| (4) | 6-point | 6-point | 6-point |



From the above comparison, the 6-point bit is evidently slower drilling under all conditions. The 4-point, although ranking first at 95 lb. pressure, under ordinary conditions would drill less rapidly than either the Carr or Z. In comparison with the others, the Z bit apparently increases in cutting speed as the pressure is decreased. This would indicate that the Z bit would be quite efficient in soft rock or at lower pressures,

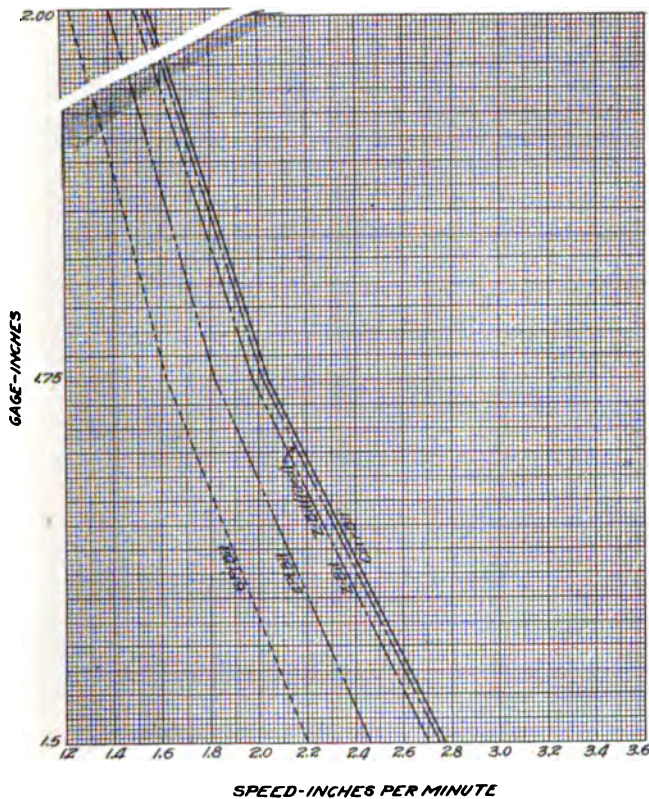


FIG. 7.—RELATIVE CUTTING SPEEDS OF 6-PT., 4-PT., Z, Z BIT (FILED) AND CARR BITS AT 85 LB. PRESSURE.

but it is evidently not adapted to extreme high pressures in hard rock. From the standpoint of cutting speed, the Carr and Z bits are the most efficient.

#### *Loss of Gage*

The average loss in gage per inch as shown from Tables 2 and 3, was as follows:

|         |         |       |
|---------|---------|-------|
| 4-point | seconds | 0.010 |
| 4-point | thirds  | 0.007 |
| 6-point | seconds | 0.010 |

|          |                 |       |
|----------|-----------------|-------|
| 6-point  | thirds          | 0.007 |
| Z bit    | thirds          | 0.004 |
| Carr bit | seconds         | 0.003 |
| Carr bit | thirds          | 0.003 |
| 4-point  | 5° taper thirds | 0.004 |
| Z bit    | filed           | 0.003 |

This loss is almost constant at all pressures. It will be seen from these figures that the loss in gage with 4- and 6-point bits is considerably more

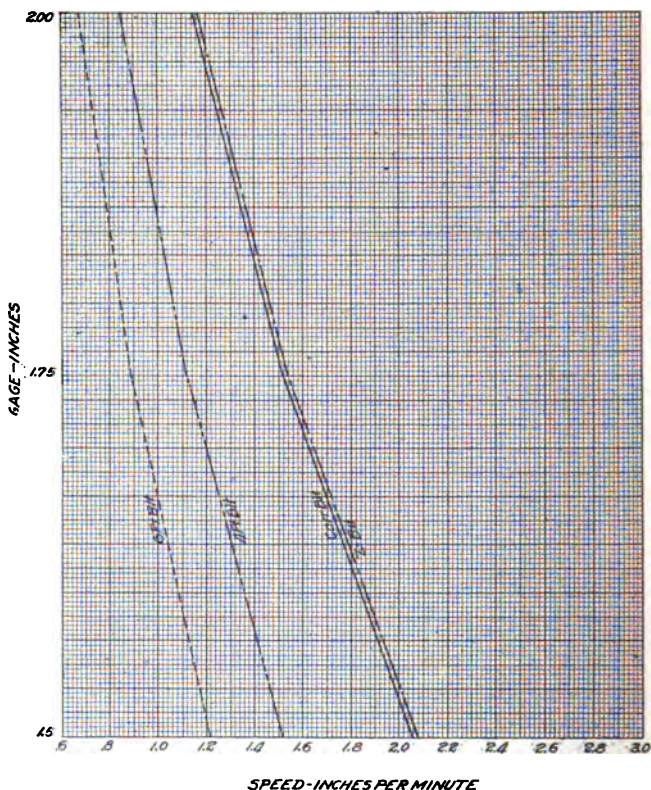


FIG. 8.—RELATIVE CUTTING SPEEDS OF 6-PT., 4-PT., CARR AND Z BITS AT 70 LB. PRESSURE.

than with Z or Carr bits, and that the 5° taper on a 4-point bit greatly diminishes the loss of gage.

From the standpoint of loss of gage, the Carr, Z bit and 4-point bit with a 5° taper, are superior to others. This factor is one that is often overlooked, and the great advantage of using a bit that loses little in gage is not generally considered. As an example the following calculation has been made:

#### *Drilling Time*

Drilling time for 6-ft. hole in granite, 1½-in. diameter at bottom. Length of changes, 1 ft. Difference in gages, 1/16 in. with Carr and Z bits, 1/8 in. with 4-point and 6-point bits.

## CARR

| Length of Steel            | 1 Ft.               | 2 Ft.           | 3 Ft.               | 4 Ft.           | 5 Ft.           | 6 Ft.           |
|----------------------------|---------------------|-----------------|---------------------|-----------------|-----------------|-----------------|
| Gage in sixteenths.....    | $1\frac{1}{2}_{16}$ | $1\frac{3}{16}$ | $1\frac{1}{2}_{16}$ | $1\frac{1}{16}$ | $1\frac{1}{16}$ | $1\frac{1}{16}$ |
| Gage in tenths.....        | 1.875               | 1.812           | 1.750               | 1.688           | 1.625           | 1.562           |
| Distance per minute.....   | 1.771               | 1.897           | 2.034               | 2.186           | 2.352           | 2.554           |
| Minutes for each foot..... | 7.340               | 6.270           | 5.890               | 5.490           | 5.102           | 4.700           |

Time 34 min. 49 sec. for 6 ft.

## Z Bit

|                            |       |       |       |       |       |       |
|----------------------------|-------|-------|-------|-------|-------|-------|
| Gage.....                  | 1.875 | 1.812 | 1.750 | 1.688 | 1.625 | 1.562 |
| Distance per minute.....   | 1.723 | 1.839 | 1.978 | 2.126 | 2.295 | 2.484 |
| Minutes for each foot..... | 6.960 | 6.530 | 6.070 | 5.640 | 5.230 |       |

Time 35 min. 16 sec. for 6 ft.

## 4-POINT

|                          |                |                |       |                |                |                |
|--------------------------|----------------|----------------|-------|----------------|----------------|----------------|
| Gage.....                | $2\frac{1}{4}$ | $2\frac{1}{8}$ | 2     | $1\frac{7}{8}$ | $1\frac{3}{4}$ | $1\frac{5}{8}$ |
| Distance per minute..... | 1.009          | 1.230          | 1.389 | 1.580          | 1.814          | 2.104          |
| Minutes per foot.....    | 11.890         | 9.760          | 8.640 | 7.590          | 6.620          | 5.700          |

Time 50 min. 12 sec.

## 6-POINT

|                          |                |                |       |                |                |                |
|--------------------------|----------------|----------------|-------|----------------|----------------|----------------|
| Gage.....                | $2\frac{1}{4}$ | $2\frac{1}{8}$ | 2     | $1\frac{7}{8}$ | $1\frac{3}{4}$ | $1\frac{5}{8}$ |
| Distance per minute..... | 0.976          | 1.092          | 1.235 | 1.405          | 1.613          | 1.867          |
| Minutes per foot.....    | 12.295         | 11.000         | 9.700 | 8.540          | 7.440          | 6.430          |

Time 55 min. 24 sec. 6-ft. hole.

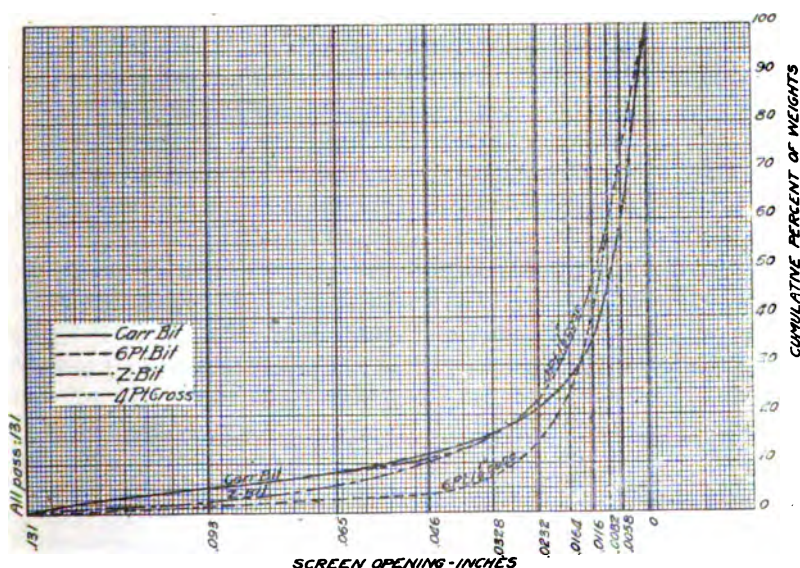


FIG. 9.—CUMULATIVE DIRECT DIAGRAM OF SCREEN ANALYSIS ON CUTTINGS AT 95 LB PRESSURE.

From these calculations the Carr and Z bits apparently save one-third in drilling time over the 4- and 6-point bits. This is due more to the fact that smaller gages can be used than to greater cutting speed. The calculations were based on actual cutting speeds and loss in gage shown in tests.

### *Breakage and Ease of Sharpening and Tempering*

There is little difference between the 4-point and 6-point bits in respect to ease of sharpening and tempering.

The Z bit is more easily made, but it is very difficult to temper so that it will stand up under high pressure. This is on account of the weakness of the outer cutting edges.

The Carr bit is by far the easiest of all to make and temper, and can be tempered much harder than others.

### SUMMARY

1. The results in down holes indicate that the cutting speed varies inversely as the square of the diameter, at least for smaller gages.

2. Drilling speed increases almost uniformly with increase in pressure, as shown in Fig. 2. A pressure of about 85 lb. per square inch seems to be best adapted to all bits for drilling in rock of the hardness of that used in the tests.

3. Speed of drilling seems to be proportional to the coarseness of the cuttings as shown by screen analysis and study of the bottoms of the drill holes.

4. Taking into consideration its cutting qualities, loss in gage, ease of making and tempering, the Carr bit seems to be far superior to all others, except possibly the Z when used at low pressures.

The Z bit at low pressures and probably in soft rock would equal if not surpass the Carr bit in cutting speed, but on account of the difficulty in its making and tempering, it is doubtful whether it would be as desirable under any conditions as the Carr.

For exceedingly high pressures in very hard rock, the 4-point bit made with a 5° taper on the wings seems to be superior to all others.

The 6-point bit apparently has little to recommend it under any circumstances, although it is convenient to use in starting holes.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Geologic Structure in the Cushing Oil and Gas Field, Oklahoma\*

BY CARL H. BEAL,† WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

### *Introduction*

DURING the latter part of 1915 and the first half of 1916, the writer held the position of geologist in connection with the conservation work instituted by the U. S. Bureau of Mines, on oil and gas land belonging to the Indians of the Five Civilized Tribes in Oklahoma. The duty of the geologist was to correlate formations containing oil, gas, and water, and to determine the relations of these substances in the different formations so that the conservation agents might carry on their work most intelligently. On account of the rapid development of the Cushing field, which lies in the western part of the Creek Nation, much work was done there by the inspection force.

In the course of the geologic studies a large amount of information was collected, which, on analysis, has disclosed some interesting facts worthy of publication. The report, of which the following remarks are an abstract, has been prepared and submitted to the U. S. Geological Survey for publication. This report is now (June, 1917) in the press and will doubtless be issued in a few weeks.

### *Methods Used*

In addition to the structure maps showing the folding of the Pawhuska limestone, which crops out in several places in the Cushing field, separate structure maps of the three most important productive sands—the Layton (Fig. 2), Wheeler, and Bartlesville—have been prepared. These structure maps were prepared by obtaining the logs of nearly all the wells

\* Abstract of *Bulletin* No. 658 to be issued shortly by the U. S. Geological Survey in cooperation with the U. S. Bureau of Mines.

Published to afford opportunity for discussion at the St. Louis meeting in October. In this brief abstract only the more important data and conclusions are given. Published by permission of the Director of the U. S. Bureau of Mines and the Director of the U. S. Geological Survey.

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drilled in the Cushing field, by subtracting the surface elevations from the depth of each sand given in the log of that well, to obtain the elevation above or the depth below sea level of that sand. Inasmuch as the accuracy of structure maps depends upon the accuracy of the elevations and of the logs, the maps prepared cannot, of course, be considered absolutely accurate, but in general they were sufficiently accurate to serve as an excellent working basis in the analysis of the information collected.

In an attempt to prepare a convergence map showing the increase in the thickness between the Layton and Bartlesville sands, the intervals between these two formations, as indicated by the available logs, were plotted on a map, but as the local irregularities were so great that it was impossible to construct a convergence map, the map which accompanies the more detailed report shows only certain groups of the same or similar intervals. For instance, all wells showing an interval of less than 1,050 ft. (320 m.) were separated from those in which the interval was between 1,050 and 1,100 ft., which in turn were separated from the areas where the interval was between 1,100 and 1,150 ft., and so on.

In studying the oil, gas, and water bodies in each sand, maps showing their relations were prepared by plotting the content of each sand as recorded in the logs collected. In studying the water conditions, the depths below sea level at which water was encountered in certain wells were plotted on a map. This map showed that the depths were very diverse, a fact that led to a detailed study, by the use of contours on the surface of the edge water, of the actual position of the water backing up the oil and gas.

### *Summary*

The work done in the Cushing field has consisted in the study and analysis of the information collected in the course of the inspection of the oil and gas land and the following principal facts have been disclosed as a result:

1. The folding of the formations in the Cushing field usually becomes greater with increase of depth, and there are many marked differences in structure among the Layton, Wheeler, and Bartlesville sands, and the surface beds.
2. The interval between the Layton and the Bartlesville sands is generally greater around the edges of the anticlines than on their crests.
3. The distribution of the bodies of oil, gas and water, indicates that the source of the oil lay west or northwest of the Cushing field.
4. In general, the oil area in an elongated dome where folding is simple extends farther down on the long axis of the anticline or dome than it does on the steeper sides; or, in other words, the area beneath which a sand contains water only extends higher on the steep flanks of an elongated dome than at the ends where the beds are more gently tilted.

5. The water surfaces on which the oil and gas rest in the different sands are not level, but are inclined away from the center of the anticlinal folds. This inclination does not conform with but is usually less than the dip of the bed in which it occurs.

### *Development*

The Cushing field, the most productive light-oil field in the world, embraces nearly 35 square miles (90.6 sq. km.) of productive territory, in which have been drilled about 2,500 wells ranging in depth from 1,200 to nearly 3,000 ft. (365 to 914 m.) (Fig. 1). Oil was first discovered in March, 1912, when C. B. Shaffer and others drilled well No. 1, on the Annie Jones (F. M. Wheeler) farm in the NW.  $\frac{1}{4}$  Sec. 31, T. 18 N., R. 7 E., about 1 mile north of the present site of Drumright. This well was drilled to the Wheeler sand and for more than a year and a half this sand produced the entire oil output of the field. In December, 1913, however, the Prairie Oil and Gas Co. completed the first well to the Bartlesville sand in Sec. 3, T. 17 N., R. 7 E. Development has been extremely rapid, especially since the discovery of oil and gas in the Bartlesville sand, and at one time the daily production of the field reached more than 300,000 bbl. of oil. The total marketed output of all the oil sands in the Cushing field has been more than 200,000,000 bbl.

### *Stratigraphy*

This investigation has been only in the nature of a study of underground conditions with a view to determining the relation of the contents of the more important oil- and gas-producing sands, and very little stratigraphic work was done. The rocks of the area, according to Buttram,<sup>1</sup> are exclusively sedimentary and, except the "terrace" sands and the alluvial deposits, are all of late Pennsylvanian age.

In this part of Oklahoma the rocks dip in general to the west, the base of the Pennsylvanian series cropping out in northeastern Oklahoma in the western foothills of the Ozark Mountains. The Pennsylvanian formations at the surface become successively younger as the Cushing field is approached and are hidden from view about 12 miles west of the field by the overlying Permian series. The exact line between the Pennsylvanian and the Permian series is uncertain, but in accordance with the provisional current usage of the Geological Survey it has been drawn at the base of the Cottonwood limestone, which is about 50 ft. above the Neva limestone. It should perhaps be drawn as low as the Neva limestone, or possibly, as some writers think, at the base of the Elmdale,

<sup>1</sup> Frank Buttram: The Cushing Oil and Gas Field, Oklahoma. *Oklahoma Geological Survey, Bulletin* No. 18 (1914).

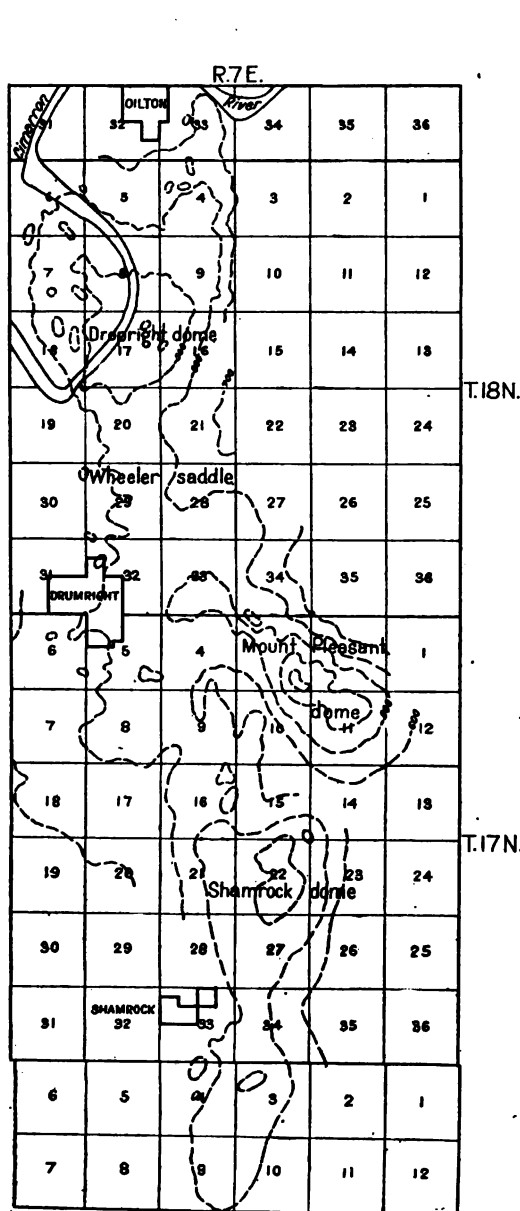


FIG. 1.

FIG. 1.—SKETCH MAP OF CUSHING FIELD SHOWING GENERALIZED STRUCTURE OF LAYTON SAND AND LOCATION OF PRINCIPAL STRUCTURAL FEATURES. CONTOUR INTERVAL IS 100 FT.

FIG. 2.—COMPOSITE COLUMNAR SECTION SHOWING RELATIVE VERTICAL POSITIONS OF PRODUCTIVE OIL SANDS IN THE CUSHING FIELD AND SOME OF THE BETTER KNOWN FORMATIONS IN KANSAS AND OKLAHOMA.

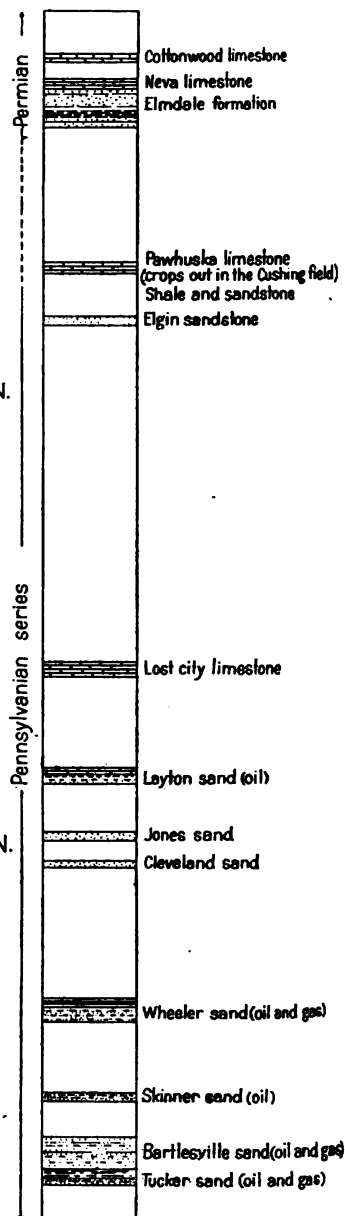


FIG. 2.

which embraces 130 ft. of sediments below the Neva limestone of the Kansas section (Fig. 2). This limestone forms an escarpment just west of the town of Cushing, Oklahoma, 12 miles west of the oil field.

The most important outcropping formation in the Cushing field is a prominent limestone, which according to Buttram is probably equivalent to the Pawhuska limestone of northern Oklahoma. Later investigations made by the U. S. Geological Survey in regions northeast of the Cushing field indicate a necessity for some nomenclatural revisions, which, however, are not yet completed. It is even somewhat probable that the limestone called by Buttram the Pawhuska and used in this report as a key horizon may not be the same as that conspicuously exposed near Pawhuska.

In the Cushing field the drill penetrates a series of alternating sandstones, shales, and limestones. The limestones apparently become less prominent and the sandstones more prominent toward the south end of the field near Shamrock. According to Buttram,<sup>2</sup> the Pawhuska limestone near the center of the field in the vicinity of Drumright lies about 2,340 ft. (713 m.) above the Wheeler sand. The Bartlesville sand in the Cushing field is supposed to be equivalent to the Bartlesville sand of the fields of northeastern Oklahoma, and lies near the base of the Pennsylvanian series.

#### *Description of the Productive Sands*

In the Cushing field, oil is being produced from six different sands—the Layton, Jones, Wheeler, Skinner, Bartlesville, and Tucker.

The Layton sand is found at depths ranging from about 1,200 to more than 1,500 ft. (365 to 457 m.), the depth depending on the locality. It is productive of oil principally in the northern part of the Cushing field, in the district south and east of Drumright, and in an area a few miles south of Shamrock. Generally it underlies a hard limestone, 10 to 20 ft. thick, called by the drillers the Layton lime, in contrast to the Layton sand, which is a soft sandstone not fully saturated with oil. The top of the sand is at many places barren and the "pay" sometimes lies in streaks, a condition probably due to differences in porosity and to intraformational barriers caused by cross-bedding. The maximum thickness of the sand reported is about 100 ft., and the average of many reports is about 50 ft. At a few isolated points no Layton sand has been found. The sand is coarse-grained, porous, and comparatively soft, and is fairly uniform in texture and porosity. In the Cushing field about 14 square miles of this sand produced oil and 12 square miles when first drilled into carried much gas.

The Jones sand lies about 200 ft. below the Layton sand and pro-

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<sup>2</sup> Frank Buttram: The Cushing Oil and Gas Field, Oklahoma. *Oklahoma Geological Survey, Bulletin No. 18* (1914), 43.

duces oil in commercial quantities only in a small area on the south side of the dome in the north part of the field, although it contains a little oil and gas at many other localities. The sand at a few places is as much as 50 ft. thick, although the average thickness lies between 15 and 35 ft.

About 100 ft. below the Jones sand is the Cleveland sand, from which, so far as the writer knows, oil has never been produced in commercial quantities, although, like the Jones sand, it contains some oil and gas at many localities. The Cleveland sand is thinner than the Jones sand and is not reported in some logs. Its thickness ranges from a few feet to 30 ft.

From 600 to 900 ft. (183 to 274 m.) below the Layton sand is the next commercially productive formation, known as the Wheeler sand, named from the Wheeler farm a short distance northeast of Drumright, where it was penetrated by the first well drilled in the Cushing field. This sand is one of the most uniform in the field in thickness. It includes the overlying "Wheeler lime," from which it is separated by a shale "break," and ranges in thickness from 50 to about 100 ft. The lower sandy member is correlated by the drillers with the "Oswego lime" of northeastern Oklahoma and southeastern Kansas. It is a coarse-grained, brownish limestone that includes porous or sandy layers which contain the oil. The part below the shale "break" is more porous than the part above it and comprises about half the formation. At some places the limestone above the shale "break" carries gas in commercial quantities and at others, on the sides of the folds, it carries water. The shale "break" between the two members ranges in thickness from 5 to 25 ft. In the Cushing field about 11 square miles of the Wheeler sand produced oil and about 21 square miles produced gas, exclusively. The Wheeler sand, like the Layton and Bartlesville sands, is spotted.

The Skinner sand lies 250 to 400 ft. below the Wheeler sand and locally produces oil in the northern part of the Cushing field and at a few places near the center of T. 17 N., R. 7 E., southeast of Drumright.

From 350 to 550 ft. (106 to 167 m.) below the Wheeler sand lies the Bartlesville sand, the most productive oil sand in the Cushing field. This sand ranges in thickness from a few inches to about 200 ft. and in porosity from compact brown shale to lenses of porous brown coarse-grained sandstone. In different wells it varies greatly in thickness, texture, and porosity and in its content of oil, gas, or water. In some wells in the north part of the field it attains a thickness of over 200 ft., and one part of the sand may be "dry," another part may carry great volumes of gas under tremendous pressure, still another part may furnish great quantities of oil and others may be filled with salt water. These so-called "streaks" are probably due to differences in porosity and apparently occur in no regular order, salt water under great pressure being found in some wells above excellent oil "pay," below which more water may be

found and below that more "pay." Notwithstanding these facts the sand has been immensely productive, though its yield has not been so great as might be expected from a body of sand so thick, for the real "pay" may form only a small percentage of the total thickness of the sand. The total oil and gas area in the Bartlesville sand is about 20 square miles, of which but 2 square miles carry gas only.

The Tucker sand lies from a few feet to about 200 ft. below the Bartlesville sand, and is thought by some to be a part of that sand. The principal area in which oil is produced from the Tucker sand lies near Drumright and is not large. This sand is uniform in porosity, medium-grained, and blue or bluish-green, and although its average thickness is perhaps less than that of the Bartlesville, not enough wells have been drilled through it to determine this question. Many of the logs show that the Tucker sand is separated from the Bartlesville sand by a thin bed of green shale, which is recognized by the drillers as a "marker."

### *Structure of the Cushing Field*

The dominant structural feature in the Cushing field is a broad north-south anticlinal fold along whose axis there are domes and along whose sides there are many subsidiary folds and irregularities. The structure contours of the three oil sands are very irregular and differ locally from the contours of the Pawhuska limestone, although the general structure axes are practically coincident. Each sand in the field exhibits small irregularities that apparently bear no definite vertical relation to each other. A detailed description of the folding of the various sands contoured and of the surface beds may be found in the forthcoming bulletin on this subject.<sup>3</sup>

Through the construction of the contour maps showing the folding of the three oil sands, it has been possible to determine in detail the differences in folding among these beds and the surface beds. The amount of folding in different directions from the crests of various domes and anticlines is given in Table 1.

In practically every case where the dip has been measured, folding becomes greater with depth. The greatest difference in folding usually occurs between the surface beds and the Layton sand.

The possible causes for the irregularities observed may be any one, or a combination of two or more, of the following:

1. The difference in resistance to compression of the hard and soft beds of which the formations in the Cushing field are composed.
2. The lenticular form of the Bartlesville sand.

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<sup>3</sup> Carl H. Beal: *Geologic Structure in the Cushing Oil and Gas Field, Oklahoma, and Its Relation to the Oil, Gas, and Water. U. S. Geological Survey, Bulletin No. 658 (1917).*

3. One or more unconformities between the surface beds and the Bartlesville sand.
4. Folding during deposition.
5. Cross-folding.

TABLE 1.—*Elevation in Feet of Highest Contour on the Surface Beds, the Layton Sand, the Wheeler Sand, and the Bartlesville Sand, on the Crests of Four Folds in the Cushing Field; also the Dip of These Beds in Different Directions from the Crests of the Folds*

(+, above sea level; —, below sea level)

|   | Dropright Dome |  |                        |                      | Shamrock Dome |                        |                      |
|---|----------------|--|------------------------|----------------------|---------------|------------------------|----------------------|
|   | Crest          | 2½ miles N. E. of crest along anti-cline | 1½ miles west of crest | ¾ mile east of crest | Crest         | 1½ miles west of crest | ¾ mile east of crest |
| Surface beds (Pawhuska limestone) ..... | +1,050         | 100                                      | 150                    | 125                  | +1,125        | 135                    | 60                   |
| Layton sand .....                       | —400           | 225                                      | 175                    | *250                 | —325          | 325                    | 250                  |
| Wheeler sand .....                      | —1,075         | 225                                      | 200                    | *350                 | —1,150        | 175                    | 200                  |
| Bartlesville sand .....                 | —1,525         | 175                                      | 200                    | 200                  | —1,550        | 325                    | 250                  |

|   | Mount Pleasant Dome |                        |                       |                            |                           | Anticline in Northern Part of T. 16 N., R. 7 E |                      |
|---|---------------------|------------------------|-----------------------|----------------------------|---------------------------|--|----------------------|
|   | Crest               | 2½ miles west of crest | 4 miles west of crest | ¾ mile south-west of crest | ¾ mile northeast of crest | Crest  | ¾ mile east of crest |
| Surface beds (Pawhuska limestone) ..... | +1,100              | 150                    | 225                   | 75                         | 75                        | +1,050   | 50                   |
| Layton sand .....                       | —275                | 300                    | 425                   | 325                        | 325                       | —400   | 75                   |
| Wheeler sand .....                      | —1,100              | 350                    | 475                   | 325                        | 400                       | —1,325   | 50                   |
| Bartlesville sand .....                 | —1,450              | 350                    | †475                  | 325                        | 400                       | —1,700   | 100                  |

\* Part of vertical distance estimated.

† Part of horizontal and vertical distances estimated.

### *The Interval Between the Layton and Bartlesville Sands*

The well in which is found the smallest interval between the Layton and Bartlesville sands lies on the west side of the Dropright dome in the northern part of the field; the interval here is 945 ft. (288 m.). The largest interval observed is 1,366 ft. (416 m.) and lies in the southern part of the field. The gradations between these limits is irregular locally, but the increase from north to south is very noticeable in general. The map prepared showing the locations of areas where wells had certain intervals shows plainly that a definite relation exists between the folds and the thickness of the beds between these two sands, the interval being generally less on the crests of the folds. This relation would naturally be



expected if the Bartlesville sand were more steeply folded than the Layton sand, and to arrive at a conclusion as to the cause of the differences in interval, it has been necessary to study the causes for the difference in folding of the various oil sands. The possible causes for the differences in folding have been enumerated above. It must not be thought, however, that the general increase in the Layton-Bartlesville interval from north to south can also be attributed to differences in folding. This increase is undoubtedly due to the conditions under which the formations between the Layton and Bartlesville sands were deposited.

### *The Direction of Migration of Oil and Gas*

As a result of the study of the relation of the oil, gas, and water, it was observed that the more important oil areas in each sand lay on the west side of the Cushing field, and gas in many cases had been forced to the east side of the structure, sometimes as far as the water line.

The hydraulic theory of the accumulation of oil and gas is based on the assumption that the hydrocarbons were concentrated by bodies of water that moved through the sand, and although the determination of the direction from which the Cushing field derived its oil and gas was not the primary object of the investigation, it was evident from the distribution of the oil and gas bodies that the hydrocarbons in this region probably migrated from the west or northwest.

### *The Relation of Oil and Gas to Structure*

The maps prepared show that the areas containing only gas conform closely to the higher parts of the anticlines and domes, and the oil areas lie farther down on the sides of the same structures. The areas overlap in each sand and have narrow irregular strips in common, where both oil and gas were originally found.

In connection with this phase of the subject, a study was made of the initial productions of the oil wells. The initial productions of a large number of wells drilled to the three sands studied were plotted on the map showing the distribution of oil and gas, and lines drawn to outline the areas where wells had certain initial productions. Representative areas of each sand show that most of the places of greater initial productions were on domes or anticlines, or that they correspond, in part at least, to areas on the sides of the folds, where both oil and gas are found in the same sand. For instance, some of the more important oil areas in the Layton and Wheeler sands lie down on the west side of the anticlines whose higher parts are completely filled with gas, and, in one noteworthy instance, the most prolific area of oil in the Layton sand conforms closely to the area wherein both oil and gas occur in this sand. On one side of

the area of high oil production, wells with small initial productions were found without any marked amount of gas, whereas on the other side gas occurs without oil.

### *Peculiarities in the Distribution of Oil and Gas*

A very interesting, and economically important fact, has been determined from a study of the distribution of the oil, gas, and water bodies in connection with the anticlines and domes in the Cushing field. In an elongated dome where folding is simple, the oil area in a sand apparently extends farther down on the long axis of the anticline or dome than it does on the steeper sides. Practically every important dome in the Cushing field, which has not been complicated by folding along its sides, shows this feature in each of the three oil and gas sands studied. The fact is shown particularly well in the Layton sand on the Dropright dome (Fig. 1) which lies in the north part of the Cushing field; on the steep west side water completely occupies the sand at a depth of 500 ft. (152 m.) below sea level, whereas on the gently dipping north side the oil extends down as far as 650 ft. (198 m.) below sea level and also completely occupies the Wheeler saddle on the south side of the Dropright dome, the lowest point of which is about 575 ft. below sea level. Another interesting occurrence is in the Wheeler sand on the Mount Pleasant dome, which lies east of the center of the field. This dome has a northwest-southeast axis and the lowest gas on the steep southwest and northeast sides is about 75 ft. higher structurally than the lowest gas on the southeastward plunging axis. No oil occurs in this sand on the Mount Pleasant dome.

Much detailed work might be profitably done, both experimentally and in the field, in connection with this interesting subject which obviously is of economic importance. If it can be determined that on domes and anticlines, folded to a certain degree of intensity, the oil is universally found lower on the long axis than on the steeper sides, this fact will be of great aid to geologists in the work of selecting lands for leasing, locating test wells, estimating the oil content, and in considering the mode of accumulation of petroleum and natural gas.

### *Inclination of Water Surfaces*

The expression "water surfaces," as here used, denotes the level or inclined plane between the oil and gas in a sand and the edge water upon which the oil and gas rest. This water surface must not be confused with the surface of the ground water, which in an undulatory region is not level, and which is defined as a surface below which the rocks are saturated with water. The water surface here referred to is a surface that is confined to one porous stratum, and has been referred to previously by other writers as the "water level."

It was decided at the outset of the inspection work in the Cushing field to collect all the available information about edge water, and if possible to reach some conclusions as to its rate and method of encroachment and its original and present level in each sand on all sides of the Cushing field. Accompanying the bulletin prepared for publication by the U. S. Geological Survey, are maps showing the contours on the water surfaces in the three different sands. The contours do not show accurately the detail of the water surfaces, and the only object in drawing them was to show that the plane of separation between the oil and the water in a given sand is not level, but dips at a gentle angle away from the center of the structure.

The most complete information has been obtained on the Layton water surfaces in the northern part of the field where the maximum dip of the water surface in the Layton sand amounts to about 100 ft. in a mile. On the west side of the Wheeler saddle which lies north of the center of the Cushing field, the contours show a western dip on the water surface of about 90 ft. in a mile. In this locality similar dips were determined on the water surface in the Wheeler sand. In the southern part of the field, in the more recent development, the water surfaces show dips of about the same amount. On account of the complexity of the composition of the Bartlesville sand the information in connection with edge water in that sand is of much less value, and the detailed studies have been confined almost entirely to the Layton and Wheeler sands.

In searching for the reason for the inclination of the water surfaces, the first point necessary to settle was, whether or not the inclination existed originally, prior to the development of the field, or was wholly the result of the extraction of oil and gas, and also, if it existed originally before the field was exploited, whether it was rendered greater by the extraction of oil and gas. If the inclination existed before the field was developed it might have a significant connection with the mode of accumulation of the oil and gas; if it was acquired since development began, a study of its causes might contribute to our knowledge of the drainage of oil sands; if it existed prior to the development and was made greater by the extraction of oil and gas, its study might afford valuable conclusions both as to the accumulation of oil and gas and as to the drainage of the sands. The methods used in determining these problems will not be discussed in this report because the discussion has been given in *U. S. Geological Survey Bulletin* No. 658, referred to above. The conclusion arrived at was that the inclination of the water surfaces in the Layton and Wheeler sands existed prior to development, and was rendered greater by the rapid extraction of oil and gas. In some places the subsequent inclination was as great or greater than the dip of the oil sands, from which it was concluded that some of the wells near the centers of the more productive areas which lie on the crests of the domes and anticlines would begin to

show water before the wells on the flanks of these folds, and after the wells on the crest had gone completely to water there might still be a small commercial deposit of oil remaining on the sides of the folds.

### *The Necessity of Similar Work*

The results obtained from this study, which was necessarily hasty and carried on without the benefit of all available information, are thought to have been well worth the effort of collecting and analyzing the information. The writer believes that much more detailed work might be done in this and in other oil fields, with very far-reaching and conclusive results, and has submitted the results of this study with the hope of stimulating similar and more detailed investigations. Undoubtedly our knowledge of the accumulation of oil and gas and the changing conditions under which they exist during development, and during the productive life of a field, is slight, and much work of this sort must be done before sufficient facts are available for the formulation of any tenable theories to account for many of the at present unexplainable phenomena encountered in oil fields.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Granite in Kansas Wells

BY PARK WRIGHT, ST. LOUIS, MO.

(St. Louis Meeting, October, 1917)

THE fact that granite has been encountered by the drill by those in search of oil and gas in Kansas is becoming more and more a matter of interest, not only to the oil producer but to everyone directly and indirectly connected with the oil business. It is, therefore, time that someone should offer an explanation for its presence and form a definite opinion based on all the data available in regard to the effect it will have on the future prospects of the area affected as probable oil- and gas-producing territory. At present there seems to be no consensus of opinion. To the writer's knowledge, only one attempt has been made to explain; by Erasmus Haworth, State Geologist of Kansas, in a paper entitled *Bulletin Number Two on Crystalline Rocks in Kansas*.

The purpose of the present paper is to offer a theory for the presence of the granite and to draw conclusions which will aid in determining the prospective oil and gas value of the area in question.

In taking up this question it will be well to consider first the tests in which granite has been reported, with a general history of each. The accompanying map (Fig. 1) shows the location of all wells described.

The first well in which granite was reported was that drilled just south of Zeandale in section 27, township 10 south, range 9 east, by a local company which had taken up a block of leases in that vicinity. At a depth of 958 ft. (291.99 m.) granite was encountered. Drilling was continued to a depth of 1,093 ft. (333 m.) in the same formation, at which depth the hole was abandoned. Another well was started 1 mile east of the first well and the same rock was found at a depth of 945 ft. (288 m.). Drilling was continued to a depth of 1,200 ft. (365.76 m.) with the bit in the same formation. Samples of the drill cuttings have been examined and they are undoubtedly crushed granite.

The next well in which granite was found was that drilled by DeLaat and Shepard, independent wildcat operators, in section 34, township 19 south, range 7 east, near the town of Elmdale. Granite was struck at 1,707 ft. (520 m.) and the hole was abandoned in the same formation at 2,525 ft. (769 m.). Mr. DeLaat watched the drilling of this well very closely and he has described to me in some detail the formation immedi-

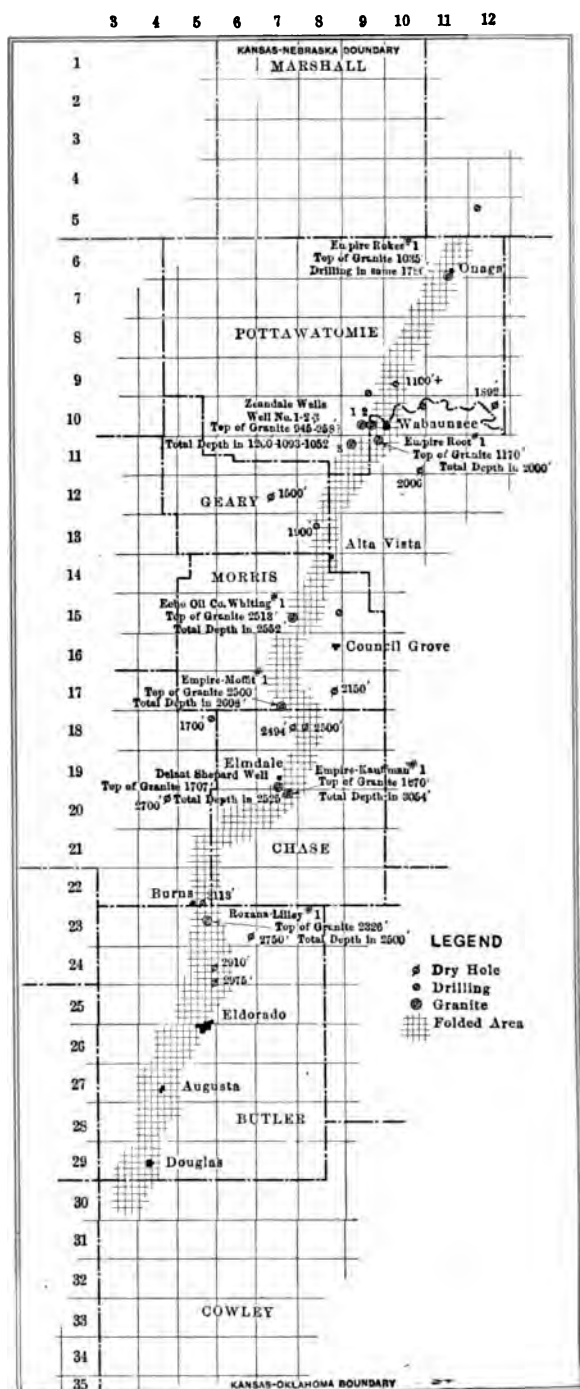


Fig. 1.

ately overlying the granite and the nature of the granite. They had about 30 ft. (9.15 m.) of cave just overlying the granite, which was composed of red mud and smooth worn slivers of limestone, the whole in an unconsolidated condition. There were no signs of contact metamorphism. The surface of the granite was badly decomposed, showing signs of weathering but unaltered below the surface.

Several wildcat wells were later started by the Empire Gas & Fuel Co., encouraged by their great success in the El Dorado and Augusta Pools. The test in section 2, township 20 south, range 7 east, was the next reported to be drilling in granite. It was encountered at 1,870 ft. (570 m.) and drilling was continued to a depth of 3,054 ft. (930.86 m.) where the whole was abandoned. A sample of the cuttings has been examined, and it is undoubtedly the same as that in the other wells.

The same company's test in section 1, township 11 south, range 9 east, got the granite at 1,170 ft. (356.6 m.) and the well was abandoned at 2,000 ft. (609.6 m.) in the same formation. Near Onaga in section 34, township 6 south, range 11 east, the Empire Gas and Fuel Co. well was drilled into granite at 1,035 ft. (315 m.) and abandoned at 1,734 ft. (528 m.).

In section 34, township 17 south, range 7 east, the Empire well drilled into granite at 2,500 ft. (762 m.) and the hole was abandoned at 2,608 ft. (794.9 m.) in the same formation.

The Echo Oil and Gas Co. in its test in section 24, township 15 south, range 7 east, reached the granite at 2,513 ft. (765.96 m.). The hole was abandoned while drilling in the same at 2,552 ft. (777.85 m.).

The Roxana test in section 14, township 23 south, range 5 east, just north of the El Dorado pool drilled into granite at 2,326 ft. (708.96 m.) and the hole was abandoned at 2,500 ft. (762 m.).

From the above data, it will be noted that there is no regularity in the depth of the granite and in no place has it been drilled through, although one test was drilled 1,184 ft. (360.88 m.) into it. At Elmdale, shallow gas was found and in the Roxanna test near Burns a small showing of oil was found in the 2,200-ft. (670.56-m.) sand. Outside of these two tests, no showings have been encountered above the granite in any of the wells drilled to it.

A very significant point is that all of the above-mentioned wells are located on or very close to the axis of an anticline and in no place has granite been found in wells drilled on either side of this continuous line of folding extending in a northeast-southwest direction across the State. Contrary to this, in several cases deep wells have been drilled favorably located on this folding which did not encounter granite. This would make it seem that if there is any direct relationship between the granite and the folding it cannot be based on the fact that it has never been found any place except on the structure without explaining the failure of wells

to reach the granite which are favorably located with reference to the structure. It is therefore necessary to account for the results of all the wells drilled in the area under consideration in order that an explanation of the presence of the granite and an estimation of its probable extent will hold good.

There are three possible modes of occurrence of the granite. We will assume that it is an igneous intrusion subsequent to the deposition of the overlying beds, a sedimentary formation composed of granitic fragments, and an ancient ridge present prior to the deposition of the overlying beds. For each assumption, the corresponding surface expressions we would expect to find will be briefly outlined so that we may eliminate those that do not satisfy the conditions as they actually exist.

If we assume an intrusion subsequent to the deposition of the overlying rocks, the chance of finding surface exposures of the granite in places where the molten rock has pushed through would be highly probable. The surface beds would show very complex folding accompanied by a larger number of faults and a general shattering of the formations due to the fact that there would be great tensile stresses produced by the pushing upward of this great mass of molten rock. The effect of contact metamorphism of the overlying beds would be evident from the drill cuttings. The surface of the granite would very probably be much finer due to the more rapid cooling of the molten mass coming into contact with the stratified rocks.

If we assume that the granite is a clastic formation, as Mr. Haworth has done in his report, classifying it as a "modified form of hard sandstone with unusually larger amounts of feldspar present, as in the Phillips County stone, cemented into this hard rock by some of nature's processes," it must be equivalent to a similar sedimentation which would be encountered in other wells over a large area which have reached the same horizon, if deposited during Pennsylvanian times. We would expect to find a more or less constant interval from certain known surface formations, especially in relatively short distances. However, if this formation is arkose, laid down prior to deposition of the Carboniferous, the above condition of constant interval would not necessarily be expected on account of the probabilities of an erosional unconformity. In either case the drill cuttings should show signs of decomposition and water-worn pebbles.

If we assume that the granite is from an ancient mountain range present prior to the deposition of the overlying beds, we would expect to find the granite body relatively extensive and continuous.

The chance of finding surface exposures would not be so great as in the case of an intrusion. The overlying beds would very probably be folded along its entire extent, forming a continuous line of structure made up of rather simple folds accompanied by little, if any, faulting. In-



stead of metamorphism at the contact of the granite with the overlying beds, we might expect an unconsolidated formation made up of the ingredients of the overlying beds together with a very small percentage of granitic material due to the abrasion of the ends of the beds against the face of the granite if folding had taken place. The surface of the granite would show signs of decomposition and weathering. Samples of the main body below the surface would show fresh, sharp angular cuttings.

Having considered the various possible modes of occurrence, the actual conditions shown by the surface rocks and those brought to light by the drill will now be presented. There is a continuous line of folding extending across the State in a northeast-southwest direction from township 6 south, range 11 east, to township 31 south, range 3 east, an air-line distance of approximately 153 miles (246.23 km.) It is very probable that this structure extends north into Nebraska, but the writer has no proof of this, not having examined that territory. The fact that the granite in all cases to date has been found only along this continuous line of structure certainly must have some significance, and if so, this continuity must be due to some cause out of the ordinary. The shape of the folding from Cottonwood Falls north is entirely different from that found in any other place not affected by the presence of granite at shallow depths. The rocks to the east of the fold dip very gently at the rate of 15 to 20 ft. to the mile into a very flat syncline which has practically no depth in comparison with the height of the anticline. From this flat syncline they rise abruptly at the rate of about 200 ft. to the mile to the crest of the anticline. After passing over the crest they gradually dip off to the west again at the rate of about 25 ft. to the mile. This shape of structure is characteristic of all folds where granite has been found except in the Roxana test on the Burns dome where it is not so pronounced and which is probably formed under slightly different conditions to be explained later. More than three-fourths of this structure has been mapped in detail and not a single fault has been found.

The overlying beds show no signs of contact metamorphism. All the formations are found regular and unchanged until a depth is reached just above the granite, where a bad cave is encountered. The amount of this cave varies in thickness. Samples of this cavy formation show an unconsolidated mass of broken pieces and long smooth slivers of limestone intermixed with shale. After passing through this the drill comes into contact with the granite, which, in most of the samples examined by the writer, shows signs of decomposition and weathering. After the surface is drilled through the cuttings show a very fresh unaltered granite. In two wells a break occurred in the granite that has been called slate. Concerning this the writer is very doubtful. The maximum depth the drill has penetrated this granite is 1,184 ft. (360.88 m.) and so far it has not been drilled through. The top has been encountered at various

depths below the Winfield limestone from 1,427 ft. (434.95 m.) at Onaga to 2,728 ft. (731 m.) at Council Grove, a range of 1,300 ft. (396 m.). At no place has the granite been found exposed at the surface.

It is plainly evident from the above data that the only logical assumption to be chosen from the three probable assumptions presented in regard to the mode of occurrence of the granite is the old range theory. There is more evidence to support the stratified rock theory than the intrusion theory, but hardly enough to balance the evidence against it. Proceeding with the assumption that a granite ridge was present prior to the deposition of the overlying beds, it is very important to ascertain just what relation the folding bears to the granite ridge, so that we may form some opinion concerning the probable extent of the granite from the surface. As stated above, in all places where granite has been found the rocks above have been folded in such a manner that the shape of the anticline and syncline formed is so different from the ordinary structure that one is led to believe that it is nothing more than the surface expression of the granite below. If, then, we can account for this extraordinary folding and its relation to the granite, it will be possible to estimate the probable extent of the granite within the reach of the drill by determining the extent of the "granite structure." In order to do this it will be necessary to work out a satisfactory theory for the various stages of development of an anticline localized by a granite ridge present prior to the deposition of the overlying beds.

In taking up this problem we will consider five stages of development. The first stage may represent the granite ridge present prior to the deposition of the overlying beds. During the second stage, deposition takes place and the beds are in the process of consolidation. During the third stage, a slight arch is formed in the beds, due to the differential consolidation of the beds under the increased load of the overlying strata on either side of the granite ridge. During the fourth stage, this arch gradually becomes more pronounced with the first effects of the lateral pressure from the east caused by the readjustment of the beds after the Ozark uplift, which seems to be the cause of all the folding along the westward dipping monocline over the greater part of Kansas. During the fifth and present stage, added pressure has caused the ends of the beds in contact with the east face of the granite and those extending over the top to slip along the face, leaving the products of the abrasion, composed of smooth slivers of limestone mixed with the shale, along the contact between the overlying beds and the granite. This condition is reasonable enough when one considers the fact that the strata against the side of the barrier which offers resistance to the lateral pressure will be most affected and will show the result of this resistance in the way suggested. The surface expression will be a sharp east dip. On the other hand, the west flank will be practically unaffected because the difference in resistance to the lateral thrust

above the granite would be so great that there would be a tendency for beds to gently "roll over" the crest and dip off gently to the west. This jamming of the beds against the granite buttress will undoubtedly be accompanied by a compression and flowage of the intervening shales which might have otherwise caused an overturned fold. The lack of faulting is probably due to the easing up of the early thrusting by the initial dip caused by the differential consolidation of the beds. The final result of these forces acting in this way will be the formation of a fold with a broad syncline, a strong east dip and a very gentle west dip just beyond the crest which is the condition existing in the folds where the granite has been found.

From the same line of thought it is reasonable to suppose that within certain limits the high places along the granite ridge will have some direct effect on the general shape and height of the structure of the overlying rocks. We would therefore expect to find the granite at the shallowest depths on the highest parts of the domes. This principle, of course, is of no value in estimating the probable depth in a proposed well. It merely serves to determine the relative chances a well has of encountering granite, given its location on the structure.

It is therefore quite possible to fail to get the granite in two wells along the axis of an anticline and encounter it at a shallower depth on the top of the dome, as in the case of the Roxana test on the Burns dome. One well to the north was drilled to 3,113 ft. (948.84 m.) and another just south to 2,910 ft. (887 m.). Both wells failed to reach the granite, yet the Roxana well drilled into it at a depth of 2,326 ft. (709 m.). This seems to show that the granite under the Burns dome comes to a high peak, causing a corresponding high place in the structure.

An examination of the shape of the folding from the south line of township 24 south, range 5 east, southwest to the end, shows none of the characteristics of the folding where granite has been found at shallow depths. It is impossible to say whether the granite extends south under this structure at deep depths, because in either case it would not be expressed in the surface structure. Moreover, this long line of folding would not be cut off sharply at the end of the granite ridge but would extend beyond the end some distance and gradually die out when it reached a point where the forces tending to continue the folding along the line localized by the granite were insufficient. The effect of this long line of folding would have a tendency to localize folds even beyond this, which probably accounts for the Blackwell folding in Oklahoma.

With these conditions existing, it is evident that the value of the granite structure for oil and gas is materially diminished because the granite has replaced large portions of the strata which ordinarily would be available as reservoirs from which oil and gas might be produced. Inasmuch as it is impossible to tell in advance of the drill the extent of this

replacement, an element of uncertainty enters which increases the unfavorable chances and therefore decreases the value of the prospect. However, in the face of all these disadvantages, all this structure should not be condemned as unfavorable. The accumulating advantages offered by the anticline may be diminished to some extent but not destroyed. It is quite possible to find productive areas along this granite structure where the conditions are favorable, *i.e.*, where the granite is 3,000 ft. (914.4 m.) below the surface for 8 to 10 miles along the axis of the fold. Even though such a place does not exist, it is likely that there are some extensive stretches of sand at shallower depths along the fold, which have not been replaced by the granite and which have been enriched with oil and gas because of the accumulating advantages offered by the anticline.

It is therefore clear that the granite reduces the value of the structure in direct proportion to the areal extent and the depth below the surface of the oil and gas sands, which the granite replaces. It is also reasonable to suppose that the replacement of these large bodies of sand will tend to act as a disadvantage toward the action of accumulation. But, on the other hand, the chance of finding favorable conditions where oil has accumulated is good enough to warrant further development. The El Dorado and Augusta fields, although they may not be deeply underlain by this granite ridge, are examples of accumulation along this same line of folding where the conditions are ideal. It is possible that similar conditions will be found to the north, but in no place is it likely that conditions just as favorable will be found.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close December 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Estimation of Petroleum Reserves\*

BY ROBERT W. PACK,† WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

### INTRODUCTION

SOONER or later in the development of any natural resource it becomes highly desirable to know the quantity of this resource in the country as a whole, as well as of the part that is being developed, for the general course and conduct of the development work must be governed to some extent by the total supply available. An accurate estimate of the quantity of any natural resource is difficult to make; yet if one is diligent enough one can obtain a pretty accurate idea of the quantity of marketable timber, the acreage of tillable land, or the horsepower that may be developed from the streams, for these are things that may be measured. But any attempt to estimate the quantity of some mineral in the ground appears to enter the realm of imagination and it seems that the estimate can be nothing more than a blind guess. It may appear to be especially futile to attempt to determine the quantity of oil and gas that is available in the ground, for these minerals are so vagrant that even if the quantity now available should be known quite exactly, the quantity that will eventually be obtained may be much less, for certain losses in the extraction of both oil and gas are unavoidable, and just what these losses will be cannot be predicted.

Though necessarily inaccurate, estimates of such mineral reserves are, however, of distinct value, for they summarize the existing knowledge of the deposits of the mineral and thus enable those who are interested in some industry that depends in one way or another on this mineral to formulate a rational commercial policy.

Some time ago, when the price of gasoline suddenly rose in a rather spectacular manner, the writer was one of the U. S. Geological Survey geologists called upon to help to estimate the quantity of oil that could be obtained in the United States, for the fear had become widespread that perhaps the rate of increase in the price of gasoline was a measure of the rate of depletion of the supply of petroleum. It thus became his

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\* Published by permission of the Director, United States Geological Survey.

† Geologist, U. S. Geological Survey.

official duty to make some sort of an estimate of the quantity of oil that still remained available in the ground in California. In making this estimate the writer attempted to predict the future output by a study of the past output of the productive fields, for such a mode of prediction is generally by far the most satisfactory one that can be used and has been employed in one form or another a great many times. There are, however, various ways in which historical and statistical data may be used, some being best suited for one set of conditions, some for another. The writer believes that the particular methods which he used in handling the data, and which are described below, are peculiarly adapted to the conditions in California, and that they are in considerable part new.

It is the purpose of this paper (1) to outline briefly one or two of the methods that have been used by others in estimating the available supply of oil, and (2) to describe somewhat more in detail the methods used by the writer, and to state why such methods were chosen and what degree of accuracy can be attained by using them. The general method followed might be applied to any oil field, provided the necessary data are available.

### GENERAL METHODS

There are two general methods of estimating the quantity of oil and gas in the ground. The first of these to be described is what is commonly known as the saturation method; the second has been termed the production curve method. The two methods are not quite comparable, for the first is used to determine the quantity of oil in the ground, and the second the quantity of oil recoverable, assuming that the conditions governing production will continue to be about the same as they have been in the past. The two methods likewise have separate applications, for the saturation method may be used to determine roughly the quantity of oil or gas in an area that has not been tested, whereas the production curve method is applicable to a partly developed field but only indirectly to an undrilled area.

### SATURATION METHOD

The saturation method consists essentially of a determination: (1) of the volume of the strata that serve as reservoirs for the oil; (2) of the porosity of these strata—that is, of the “voids” in them—and thus of the maximum volume of the space that may be occupied by the various fluids; and (3) of the percentage of the “voids” that is occupied by oil. This method is the one that has commonly been used ever since estimates of the oil reserves of the country were first attempted. The method has recently been described in some detail by Washburne.<sup>1</sup>

The saturation method obviously involves the consideration of several

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<sup>1</sup> C. W. Washburne: *Trans.* (1915), 51, 645.

factors that cannot be measured directly. The first factor to be considered—the volume of the sand that serves as a reservoir—is the factor that may be determined most accurately, and in fields where the strata show relatively little variation in lithology and thickness over large areas, such as the Appalachian fields, it may be possible to calculate the volume with considerable accuracy from measurements made on the outcrops of the oil- or gas-bearing sand, or from those given by a few scattered wells. In a very great many areas, however, and of these the California fields are fairly typical, the formations are so lenticular and the reservoir sands so inconstant both in thickness and in lithology that no reliance can be placed on the calculations of the volume of the reservoir made in this manner, even if the points at which the measurements were made lie fairly close together.

The second factor, the porosity of the reservoir sand, is even more difficult to estimate. In making an estimate of the oil, it is clearly useless to determine the porosity of the oil-bearing sands in areas where the total volume of the strata cannot be determined with some degree of precision; but even in areas in which the thickness of the formations is fairly constant, and in which the volume of the beds acting as reservoirs may be calculated with fair precision, the porosity may vary greatly, either because of variation from place to place in the grain of the rock or because of the local introduction of a secondary cement that occupies a considerable part of the original void space of the rock; and the variation may be so great that the space which may be occupied by fluids cannot be calculated. In these areas the porosity of the oil-bearing bed as determined from samples of rock taken on the outcrop or from a few samples obtained in drill holes is apt to be very different from the average porosity of the stratum over a large area.

The third factor, the percentage of the void space that is occupied by oil, is, of course, not directly measurable, so that it is necessary to assume either that the voids are all so occupied or that some arbitrarily chosen proportion of them are.

Should all these different factors be correctly determined, the resulting figures will record the total quantity of oil in the field. Not all this oil is recoverable, however, and the figures so obtained must be multiplied by an "extraction factor" in order to obtain figures that will represent the available oil.

Even under the most favorable conditions, then, the saturation method is capable of giving only a rough approximation of the recoverable oil in any area; but rough as it is, this method has a distinct value, for it will give directly a first approximation of the maximum amount of oil to be expected from an untested or very sparsely drilled area in which the formations are fairly constant in thickness and lithology. This method is not, however, applicable to areas where oil is held in solution

cavities or in fracture zones, nor is it of any very great value in the consideration of pools that occur in the variable Tertiary strata of California.

### PRODUCTION CURVE METHODS

The production curve method is a mode of predicting the future production of an area and is based on a study of the past production of that area. The method is largely graphic, and although various modifications of it have been used all are essentially extrapolations of one or more curves that show the production or development of the field. The advantages of this method over the saturation method are: (1) it is concerned with actual production, so that the assumption of any "production factor" is unnecessary; (2) the factors that must be estimated or assumed are fewer and on the whole less liable to great variation than those that must be estimated or assumed for the saturation method.

Two general methods of using production figures are employed in estimating the reserves of oil in a given area. One uses figures showing the production per unit of time of the whole field or of a group of fields within a given area; the other concerns itself only with the production of certain small areas, which are considered as types, the results obtained in their study being applied to the field as a whole.

In support of the first method, it has been urged that a consideration of the combined production of a large number of wells and of wells distributed over a large area automatically takes account of various irregularities in the production of individual wells—of irregularities due to variation in demand, to increased drilling, and to the discovery of new pools—it being always assumed, of course, that the conditions in the future will vary about as they have varied in the past. But in the very fact that the curve incorporates these factors lies its weakness, for in the nature of things drilling cannot continue indefinitely into the future, nor is it reasonable to expect the continuous discovery of new pools. These two factors should therefore be taken into account separately, and their effect should be eliminated from the curve showing total production.

The second method—that of studying the history of a small group of wells—has an advantage in that it permits a more intensive study, but it has a disadvantage in that the data necessary for this study are much more numerous and are seldom available. Moreover, in applying to the whole field the results obtained in the study of a small part of it, it is necessary to estimate the degree in which this area represents the conditions that exist throughout the field.

#### *Method Using the Record of the Production of the Entire Field*

*Projection of Curve Showing Quantity of Oil Produced.*—Probably the simplest method of predicting the future of a field is to project the curve that represents the field production per unit of time. Such a method



will give reasonably correct results only for an old field in which the production of oil per unit of time has been constantly decreasing. When applied to the Appalachian fields it gives fairly satisfactory results. The method of representation shown in Fig. 1, in which the shaded area below the line *ABC* represents the volume of oil that has been produced to date from an old field, that has long been producing a constantly decreasing quantity of oil in a given length of time. Assuming that this rate of decrease will continue until the field is completely drained, the amount of oil now remaining in the field is represented by the unshaded area below the line *CD*.

This method of prediction is open to the criticism that it does not discount the effect of new wells but that it tacitly assumes that new wells will continue to be drilled up to the very moment that the pool is drained,

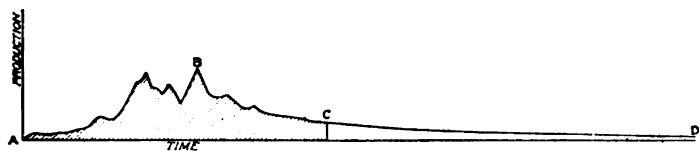


FIG. 1.—ESTIMATION OF RESERVE OF PETROLEUM BY EXTRAPOLATION OF CURVE SHOWING FIELD PRODUCTION IN UNIT TIME.

or, if the curve represents the production of a large area rather than of a single pool, that new pools will continue to be discovered up to the moment when the oil in the region is exhausted.

*Curve Showing Percentage of Decrease in Production.*—A variation of the graphic method of estimating reserves just described is to plot, not the figure showing the actual quantity of oil produced, but a figure showing the relation between this quantity and the maximum quantity of oil that the field has produced during some unit of time. The curve so obtained—the so-called percentage curve—was used by Arnold, and Fig. 2 is adapted from his plate.<sup>2</sup>

This mode of estimating oil reserves, like that just described, is applicable only to those fields that have passed the zenith of their productivity and are producing in any unit of time a constantly decreasing quantity of oil. The curve in Fig. 2 represents the life of the field after the time when the maximum yield per unit of time was obtained. It is so constructed as to show the quantity of oil obtained during each unit of time that followed the period of maximum yield, the quantities being expressed in percentages of that maximum. This method is also open to the objections that have just been raised against the method of projecting the curve that shows the total production directly, particularly to the objec-

<sup>2</sup> Ralph Arnold: The Petroleum Resources of the United States. *Economic Geology* (December, 1915), 10, 695-712.

tion that it in no way takes account of the effect of new wells. The sole object of plotting the statistics of production in this form was to obtain a curve that would serve as a type to show decline in production for any and all fields. The rate of decrease in production, however, is so different for different fields that it seems very doubtful whether a single curve can express the decline for more than a single field, or perhaps for a group of fields in rock of the same general type or of the same general geologic structure.

This method of plotting production in terms of percentages of total production is essentially an attempt to apply to the country as a whole

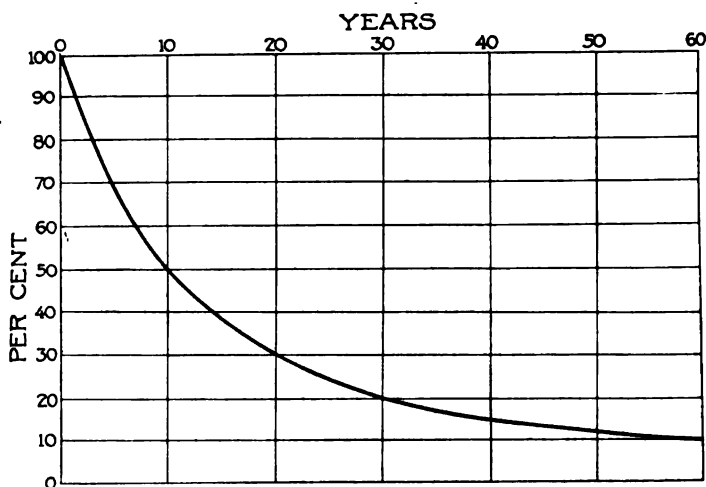


FIG. 2.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF "THEORETICAL CURVE OF FINAL DECREASE."

the methods employed by the California evaluation committee, whose work is explained below. That committee, however, concerned itself with small areas for which a mass of detailed information was available and constructed a number of curves, each showing the rate of decline for a certain particular field. An inspection of these curves, all of which were prepared for fields in California, shows that even within that State the decline is different for different fields. It is evident that similar curves constructed for fields in Oklahoma and Pennsylvania would show still greater differences. Moreover, the production curves of the evaluating committee were so constructed that the effect of continued drilling is to a measure discounted. The "theoretical curve of final decrease" used by Arnold does not in any way consider the effect of drilling new wells, but, like the method shown in Fig. 1, assumes that drilling will continue up to the very moment that the field is totally drained. This method, therefore, has less to recommend it than the method of projecting the curve showing the exact quantity of oil produced in some unit of time.

### *Method Using Record of the Production of Small Areas*

*Curve Showing Production of the Average Well.*—Some time ago, when a plan was on foot to combine certain producing properties in California, it became necessary to determine the relative value of the different properties, and especially the reserves of oil contained in each. A committee, of which M. L. Requa was the moving spirit, attempted to determine the reserves by making a detailed study of the history of the wells. The method that they followed was essentially the study of a small area in great detail, and the application of the results of this study to the field as a whole. The immediate problem was not to determine the reserve

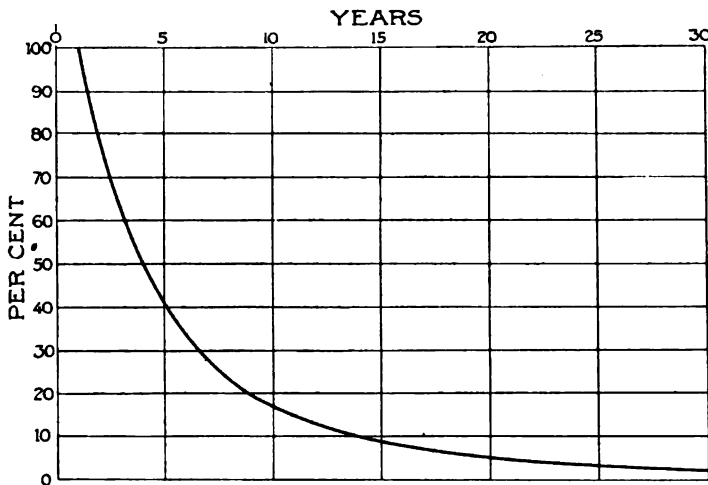


FIG. 3.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF CURVE SHOWING PRODUCTION OF AVERAGE WELL EXPRESSED IN PERCENTAGES OF PRODUCTION OF WELL DURING FIRST YEAR OF ITS LIFE.

of oil in the whole field, for that was of only incidental interest, but the amount remaining in certain parts of the field. The areas studied, however, were regarded as types of the fields or of the parts of the fields in which they lay, and the curves showing the predicted production for the small parts were applied to the whole field.

Then, too, the manner in which they used the record of the production of the wells was unique. Instead of plotting a curve that showed the rate at which oil had actually been taken from the area—that is, instead of constructing what might be termed a chronologic curve—the figures showing the production of the wells were so grouped that the quantity of oil obtained from a well during each year of its life was grouped with the quantity of oil obtained from all the other wells during the corresponding years of their lives, irrespective of the actual time at which the wells were drilled. Thus, although well B may have been

drilled 10 years later than well A, the oil that well B produced during each year of its life was grouped with the oil that well A produced 10 years before. The figures showing the production of the wells as thus grouped were then expressed in percentages of the production of the first year, and this curve was taken to represent the production of the average well in the area. To get a figure representing the quantity of oil that the average well will produce, it is necessary only to determine what the average production during the first year of their lives has been for the wells that have been drilled, and to substitute this number in the curve. The oil produced during the following years is then expressed in percentages of this number.

In order to obtain a figure representing the total production of an area, it is necessary to determine the number of wells that would be necessary to drill the area completely, and to multiply this number of wells by the figure showing the total quantity of oil obtainable from the average well, as shown by the curve. A curve constructed for one of the California fields is shown in Fig. 3.

This method has the distinct advantage over the methods hitherto described in that it attempts to determine the effect upon production of the rate at which new wells are drilled. The degree to which this discounting is really effected is, of course, dependent in large part upon the degree to which the property is drilled, for the mutual interference of wells increases as wells are multiplied.

#### METHODS USED BY THE WRITER

In casting about for some way in which to estimate the oil reserves in California, the writer, after considering the methods just described, concluded (1) that the production curve rather than the saturation method should be used in estimating the reserves in the producing fields, and (2) that the untested areas should be estimated by comparing them with the producing areas that they most closely resembled geologically, after the reserves in these areas had been estimated by the production curve method. The best way of handling the production record did not seem to be quite so clear, for none of the methods appeared quite to fit the requirements. It seemed better to use a method that considers the production of the whole field rather than one that considers only a small part of it, both because the data necessary for the detailed study of a small area are seldom available and because of the difficulty in choosing areas that might fairly represent the whole field. It appeared, however, that in dealing with the record of the total production some way should be found to discount the effect of the new wells.

#### METHODS USING FIGURES SHOWING FIELD PRODUCTION

In order to fulfil approximately these requirements, the method

described below was devised. Fig. 4 shows the application of the method to a California field:

1. The field production per unit of time was plotted and a curve, *ABCF*, similar to that in Fig. 1 was obtained.

2. Two points, *B* and *C*, which represented dates when the field production was the same, were selected. These points were as far apart as possible and corresponded to the general level of the curve about them—that is, they represented dates when the field production was fairly constant.

3. The number of the new wells drilled between the two dates represented by points *B* and *C* was then determined, this, of course, being a matter of record, and,

4. The number of new wells required to drill the field completely was then estimated from a knowledge of the geology of the field, it being

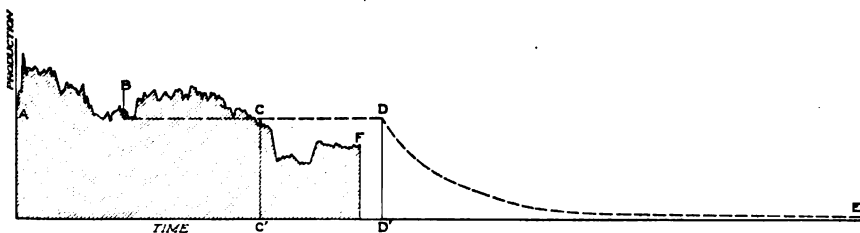


FIG. 4.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF CURVE SHOWING FIELD PRODUCTION IN UNIT TIME ASSUMING THAT DRILLING MAINTAINS PRODUCTION AT CONSTANT FIGURE.

assumed that the manner of locating wells would in the future be about the same as it had been in the past.

These factors having been determined, the supposition was made that, if new wells were drilled after the time represented by the point *C* at the same rate as they had been drilled during the period represented by the line *BC*, the production of the field would be maintained at a constant figure. This is not precisely correct, for the initial production of the wells that are drilled late in the life of the field is not nearly so great as that of the wells drilled while the field is new. The number of new wells necessary to maintain production will therefore increase as the field grows older. An idea as to what this rate of increase is in the number of wells necessary, may be obtained from a study of a cumulative curve showing the time at which new wells have been drilled and a curve showing the production of the field. The correction may be neglected in making the first approximation of the reserve of petroleum in a field, for other factors which enter into the calculation are liable to so great error that such a refinement seems insignificant, but in making a detailed study of a small area where a considerable number of reliable data are available, such a correction should be made. When the rate has been

determined at which the drilling of new wells must progress in order to keep the output of the field constant, the time at which the field would under these conditions be completely drilled is calculated. This time is represented on the diagram by the point *D*. The quantity of oil which under these conditions would be produced between the time represented by point *C* and the time represented by point *D*—the time at which the field would be completely drilled—may be calculated or may be measured directly from the diagram. After the field is completely drilled a considerable quantity of oil will still remain in the ground and the field will continue to produce at a constantly decreasing rate. The rate of decrease may, however, be estimated fairly well, for it is unaffected by the flush production of new wells.

The most satisfactory method of making this estimate is to choose some small area containing a number of fairly old wells, preferably one

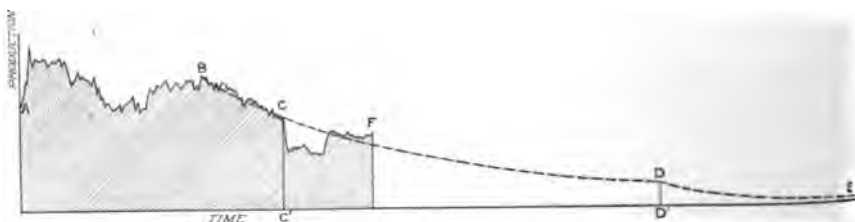


FIG. 5.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF CURVE SHOWING FIELD PRODUCTION IN UNIT TIME, ASSUMING THAT DRILLING MAINTAINS PRODUCTION AT A UNIFORMLY DECREASING RATE.

in which all the wells were drilled at about the same time, and to plot the production per unit of time of this group of wells after the date at which the last well was completed. The resulting curve may be taken as typical of the last period of the life of the average well in the field and it may be applied directly to the curve *ABCD* at the point *D*, which represents the time at which the field is completely drilled.

It may so happen that the field has produced a constantly decreasing quantity of oil since almost the first year of its life and that it is not possible to find two dates far enough apart at which the total production was the same. It may be possible, however, that for a considerable time the decrease in production was fairly constant. If so, a calculation similar to that required in the preceding method may be made, with the difference that drilling is theoretically continued at a rate sufficient to maintain not a constant production but a regular decrease similar to that which takes place during the period chosen. Then, in a manner like that just described, the date at which the field would theoretically be completely drilled may be calculated and the production up to this time estimated. The oil remaining in the field after the completion of drilling

could then, as before, be estimated by assuming that the rate of decrease of production would be like that shown by a typical group of old wells.

An application of this method is shown in Fig. 5. Here the shaded area below the line  $ABCF$  represents the volume of oil produced to date from the field. On the curve  $ABCF$  a section  $BC$  is chosen during which the decrease in production is fairly constant; the number of new wells drilled during this time is ascertained and, on the assumption that this rate of drilling will be continued from the time represented by point  $C$  until the field is completely drilled, the time at which the field will be completely drilled is calculated. This time is represented on the diagram by point  $D$ . The quantity of oil that will be produced between the time represented by point  $C$  and the time at which the field is completely drilled is represented by the area inclosed by the lines  $CD$ ,  $DD'$ ,  $D'C'$ ,  $C'C$ . The oil that is contained in the field after drilling has been completed is represented by the area lying below the line  $DE$ .

In this method, as in the preceding one, the assumption that if new wells are drilled after the time represented by the point  $C$  at the same rate as they had been drilled during the period represented by the line  $BC$ , production would decrease at the rate at which it decreased during the line  $BC$ , is not quite correct. Because of the smaller initial production of wells drilled late in the life of the field, the number of wells necessary to maintain this rate of decrease in production of the field will gradually increase as the field is drilled up. A correction similar to that suggested in the method just described may be made here.

#### METHOD USING FIGURES SHOWING FIELD PRODUCTION EXPRESSED IN TERMS OF PRODUCTION PER WELL PER UNIT OF TIME

During the first few years of the life of a field, drilling is usually so actively carried on that the quantity of oil produced in the field as a whole during a given unit of time is constantly increasing. It is obviously impossible to apply either of the methods just described to this early period in the life of a field, for they are applicable only to a period when the field production per unit of time is decreasing. For many fields, however, a fairly satisfactory estimate may be made by expressing the production of the field per unit of time in terms of the production per well per unit of time, for although the production of oil obtained during a given length of time from the field as a whole is increasing, the production of the average well during equal intervals of time is the same or is decreasing. This method of estimating the petroleum reserves is not so accurate as either of the preceding methods, because it is based upon fewer data. It should not be used, therefore, if either of the other methods is applicable.

Fig. 6 shows the application of this method. The production of the field is represented by the line  $AB$ . This curve shows a constantly increasing value for the production of the field during a unit of time, so that the methods represented by Figs. 4 and 5 are not applicable. The production per well per unit of time is therefore determined, and the curve  $CD$ , showing these values, is plotted. On the curve  $CD$  the points  $E$  and  $F$  are chosen as representing dates at which the production of the average well—the production per well per unit of time—is the same. The rate at which new wells were drilled during the time  $EF$  is ascertained and, as before, the length of time necessary to drill the field completely is

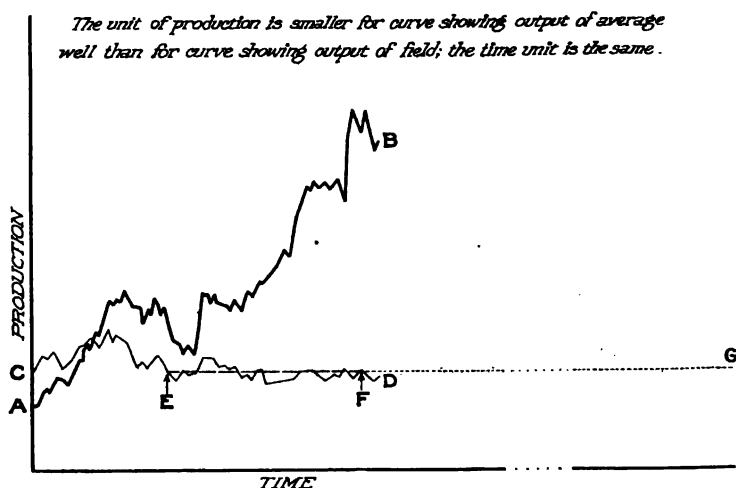


FIG. 6.—ESTIMATION OF RESERVE OF PETROLEUM BY USE OF CURVE SHOWING PRODUCTION PER WELL PER UNIT TIME ASSUMING THAT DRILLING MAINTAINS PRODUCTION OF AVERAGE WELL AT CONSTANT FIGURE.

determined, it being assumed that drilling will be continued at a rate that will maintain the production of the average well at a constant figure—a figure equal to that of the average well at the dates represented by points  $E$  and  $F$ . A first approximation of this date at which the field would be completely drilled may be obtained by assuming that production would be maintained at a constant figure if drilling be carried on after the date represented by point  $F$  at the same rate as it had been carried on during the period represented by the line  $EF$ . To be accurate, this assumption as to the rate of drilling after the time represented by point  $F$  should be corrected to take into account the smaller initial output of the wells that are drilled late in the life of the field. The time at which the field would be completely drilled is represented on the diagram (Fig. 6) by the point  $G$ . Then if

$a$  = the number of wells producing at  $E$ —the first date,



- $b$  = the number of new wells drilled during a unit of time in the period between  $E$  and  $F$ —the two dates chosen,  
 $c$  = production of the average well in a unit of time during the period  $EF$ ,  
 $d$  = the number of wells necessary to drill the field completely, (This number may be estimated from a knowledge of the geology of the field and of the field practice in spacing wells),  
 $S$  = the total amount of oil that would be produced between dates  $E$  and  $G$ .

And assuming that new wells are drilled after the time represented by the point  $F$  at the same rate as they had been drilled during the period represented by the line  $EF$ , the total quantity of oil produced by the field from the date  $E$  until the field has been completely drilled would be given by the formula:

$$S = ac + (a + b)c + (a + 2b)c + (a + 3b)c + \dots + dc$$

$$= \frac{n}{2}(ac + dc), \text{ where } n = \text{the number of terms in the series.}$$

but:

$$n = \frac{d - a}{b} + 1$$

therefore:

$$S = \frac{d - a + b}{b} (a + d)c.$$

The total quantity of oil produced before the time represented by  $E$  should be a matter of record. The oil remaining in the field after it had been completely drilled under the conditions stated may be estimated as in the two methods just described by assuming that the rate of decrease in production will be the same for the field as it is for a small group of old wells.

If the smaller initial production of the wells drilled late in the life of the field is considered and the rate of drilling of new wells corrected to show this factor, the formula given above will not apply. The curve shown in Fig. 6 may, however, be projected in a manner similar to that in which the curve in Fig. 4 was projected. This curve will show the total amount of oil that the average well will produce, from which the total reserve in the field may be calculated.

Should the field production be increasing, but the production of the average well decreasing, the size of the available reserve may be estimated by a method similar to that shown in Fig. 5, but with the difference that the production figures plotted express the production of the average well, and not of the field as a whole.

## PROBABLE DEGREE OF ACCURACY OF ESTIMATES

All the methods of estimating petroleum reserves described above involve the consideration of facts that, in the very nature of things, cannot be determined precisely, and the results obtained, therefore, cannot be precisely accurate. The results obtained by the methods that take into account the greater number of the variable factors are probably the more accurate. The writer believes that the methods he used in estimating the reserves in California express correctly the order of magnitude of the reserve, but he would not claim for them greater accuracy than that.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Geosynclines and Petroliferous Deposits

(A Contribution to the Study of the Relations between Earth Movements and Hydrocarbon Accumulations)

BY MARCEL DALY, SEATTLE, WASH.

(St. Louis Meeting, October, 1917)

In a preceding paper<sup>1</sup> the writer has pointed out some apparent relationship between the distribution, on the surface of the globe, of the known hydrocarbon deposits and the disposition of the principal zones of deformation of the earth's body (geosynclines). He wishes to present some further remarks on this subject.

Petroleum deposits, or their derivatives or descendants, are found through the whole range of sedimentary strata, from the pre-Cambrian to the Quaternary. For instance, the pre-Beltian (Shuswap) series of the Canadian Cordillera are interbedded with limestones which are sometimes rich in carbonaceous matter (Sicamous limestone); and the Beltian system of the same region contains layers of argillites and dolomitic limestones which are equally high in carbon content.<sup>2</sup> This carbon may be interpreted as the last remnant of hydrocarbons previously contained in the rocks. On the other hand, the actual formation of hydrocarbons under conditions that would eventually permit their deposition in modern sediments is conceded by some geologists.<sup>3</sup> Between these two extreme limits, bitumens are found indifferently in all sedimentaries. They constitute a continuous series, some kind of a large family, whose individual members may be entitled to a community of origin and probably represent the different stages of evolution of the primitive, parent matter. The kinship between petroleum and such substances as asphalt, ozocerite, manjak, grahamite, albertite, etc., is well known; and E. H. Cunningham-Craig has recently shown how intimately shale fields and oil fields are connected.<sup>4</sup> For this reason, no distinction is made in the present

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<sup>1</sup> The Diastrophic Theory. *Bulletin* No. 115 (July, 1916).

<sup>2</sup> *Geological Survey of Canada, Guide Book* No. 8, 124 and 134.

<sup>3</sup> Sir Boverton Redwood: *Petroleum* (1913), 1, 133-134; Dalton: *Economic Geology* (1909), 4, 620; E. T. Dumble: *Trans.* (1914), 43, 526.

<sup>4</sup> E. H. Cunningham-Craig: Kerogen and Kerogen Shales. *Journal of the Institution of Petroleum Technologists* (1916), 2, 238-273.

study between the different kinds of hydrocarbons (petroliferous deposits), and they are treated as a whole.

James D. Dana has applied the term *geosyncline* to the great earth troughs that have taken place in regions of excessive deposition, and that he considers preparatory to the formation of mountain ranges.<sup>5</sup> Subsequent writers, and especially Prof. Emile Haug, of Paris, have greatly extended the meaning of the term as well as the idea itself. Haug considers the geosynclines as zones of weakness and mobility of the earth between two stable masses (continental areas);<sup>6</sup> whereas T. C. Chamberlin regards "the crumpled tracts as lying on the border of great segments of the earth that acted essentially as units."<sup>7</sup> From a mechanical standpoint, the idea is fundamentally the same. Considered in this light, a geosyncline is no more simply the expression of a local deformation: it belongs to a general system of distortion which is world wide (network). The term geosyncline is used in this paper with this extended sense, but without any special inference as to the real nature of the intervening areas or earth segments, which is immaterial for the present discussion. Individually, geosynclines are to be considered "the sites of subsequent foldings of the strata, or, conversely, the regions of folded mountains are the sites of former geosynclines."<sup>8</sup> A geosyncline may be simple or compound, and is more generally compound.

It seems difficult to call in question the fact that the location as well as the trend of these world-wide zones of mobility have somewhat varied in the course of ages. For instance, Marcel Bertrand has shown<sup>9</sup> that three successive zones of distortion have developed in northern Europe during Paleozoic and pre-Paleozoic times, progressing from North to South and giving rise to three distinct mountain chains, which would have been preceded by as many geosynclinal belts.<sup>10</sup> On the other hand, in some regions, the general strike of the foldings seems to have remained constant during extensive periods. Thus, in British Columbia, the

<sup>5</sup> J. D. DANA: *Manual of Geology*, 4th Ed., 380, 385.

<sup>6</sup> Emile Haug: *Les Géosynclinaux et les Aires Continentales. Contribution à l'étude des transgressions et des régressions marines. Bulletin de la Société Géologique de France* (1900), 3 Ser., 28, 617-711; also *Traité de Géologie* (1908), 1.

<sup>7</sup> T. C. Chamberlin and R. D. Salisbury: *Geology*, 1, 543.

<sup>8</sup> A. W. Grabau: *Principles of Stratigraphy*, 902.

<sup>9</sup> *Bulletin de la Société Géologique de France* (1887-88), 3 Ser., 16, 576.

<sup>10</sup> M. Bertrand terms the first chain the *Huronian*. It would have extended, in pre-Cambrian times, from the Lake region of North America through the Highlands of Scotland, Norway, Sweden, Finland, and reached the extremity of Asia. Later, a second chain (the *Caledonian* chain of Suess), would have erupted further south, toward the end of the Silurian; and, finally, toward the end of the Carbonic, a third one, the *Hercynian* chain (Variscic or Armorican of Suess) would have arisen still further south, extending from southern Ireland and French Brittany, through the Vosges, the Ardennes, the Black Forest and the Hartz, to Bohemia. The Appalachian folding, which dates of this same period, would have been a link of this latter network.

Rocky Mountain formations, above the Shuswap (pre-Beltian) series, show a Cordilleran trend throughout; and from the Beltian (pre-Cambrian) to the Permian, the enormous pile of sediments involved, more than 50,000 ft. thick, present no thoroughgoing unconformity. This would indicate, at least for this part of the Pacific Geosyncline of North America, a permanency in the direction of the strains extending through the greater part of the known history of the earth. But even there a change may be detected if we trace the movements further back; for the underlying Shuswap (pre-Beltian) strikes N. 70° E., nearly at right angles to the prevalent strike of the later formations.<sup>11</sup>

Thus, in order to examine in some detail the mutual relations between geosynclinal belts and hydrocarbon (petroliferous) deposits, it would be necessary to define the different successive synclinal networks at the different periods of the earth's history and to compare them with the corresponding known petroliferous deposits of the same age. But this is a task which, at present, would be beyond our means, by reason of the lack of data. We are reduced to general comparisons, of widely separated periods of time. However incomplete such an examination may be, it offers, in the opinion of the writer, some conclusions of real import.

By far the great majority of the known petroliferous deposits of commercial value belong to the Mesozoic and to the Tertiary eras; or at least the sediments in which they are found date from these ages. Further, there seems to have been a certain degree of constancy, since the close of the Paleozoic, in the location and general trend of the principal areas of deformation of the globe: most of the Tertiary areas of deformation closely follow the trend of Mesozoic synclinals. Finally, the distribution and disposition of the geosynclinal belts of post-Paleozoic age are better known to us than those of the preceding eras. For these reasons, it has seemed advisable to the writer to investigate first the Mesozoic and post-Mesozoic periods.

The map (Fig. 1) shows the results of such an investigation. This map has been established on a Mercator projection (longitudes E. and W. of Paris). The geosynclines are those of the Mesozoic era, and are drawn according to the views of Prof. F. Haug.<sup>12</sup> The dotted part of the outlines are those which this author considers either uncertain or hypothetical. The Mesozoic and Tertiary petroliferous deposits are represented by black dots; the Paleozoic ones by shaded areas. No attempt

<sup>11</sup> Reginald A. Daly: Introduction to the Geology of the Cordillera., *Geological Survey of Canada, Guide Book No. 8*, 149 and 153. As R. A. Daly observes, this strike recalls the prevailing E. W. to N. 60° E. strikes in the pre-Cambrian rocks of Lake Superior and eastward thereof in the Canadian shield. It also recalls the Huronian system of M. Bertrand, and may indicate a western extension of this pre-Cambrian synclinal network.

<sup>12</sup> *Bulletin de la Société Géologique de France* (1900), 3 Ser., 28.

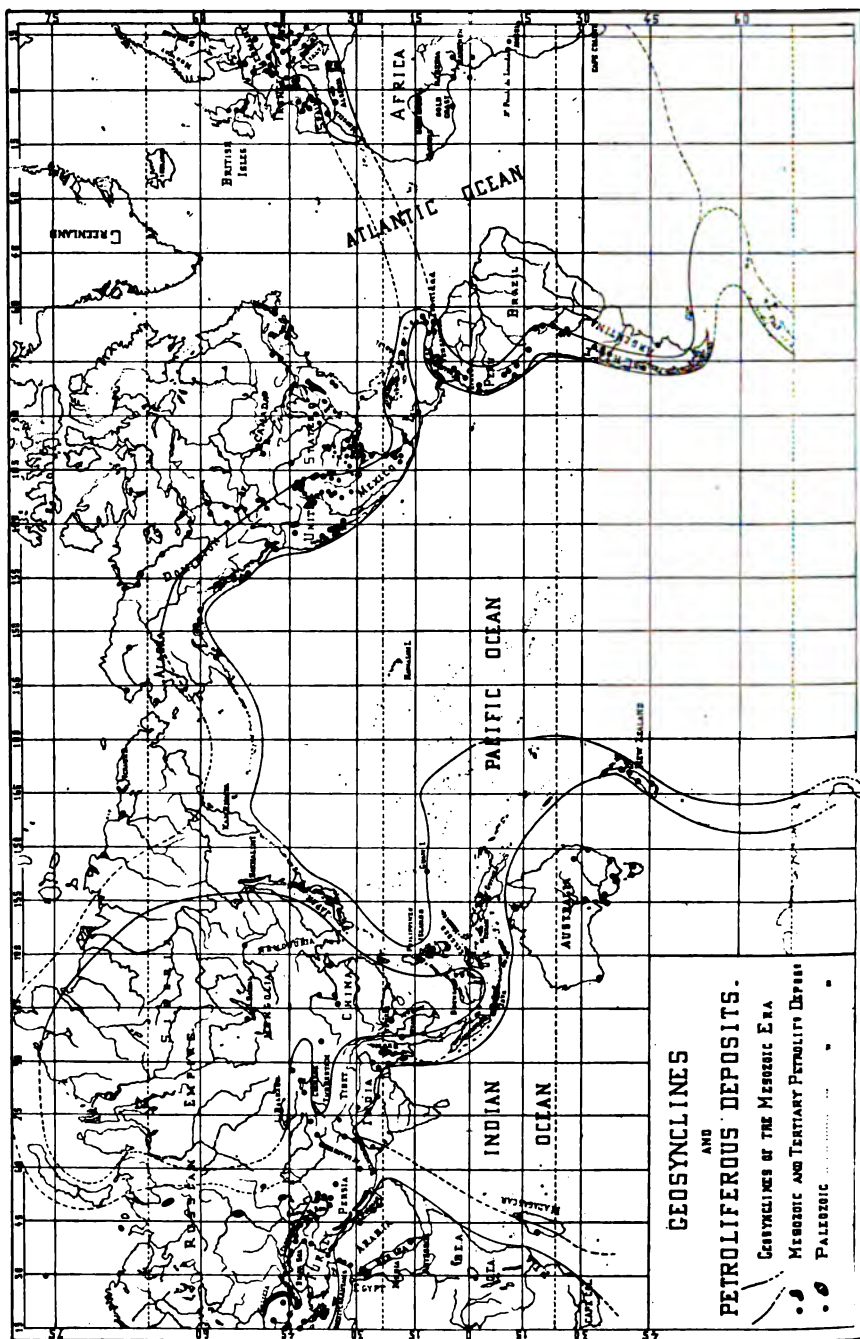


FIG. 1.

has been made to distinguish between the various hydrocarbons nor to set off the respective importance of the deposits themselves. For the location of the deposits, use has been made of the maps published by Sir Boverton Redwood in his treatise on Petroleum and of the publications of the Geological Surveys of the United States, Canada, India and others available to the writer.

The aim of this map is to present general information rather than to give data correct in every detail. The limits of a geosyncline can hardly be determined with any degree of geometric accuracy. The principal areas of dislocation and folding of a region are defined, but minor folds generally extend at a more or less greater distance outside of their assigned limits. As pointed out by De Launay,<sup>13</sup> "the dislocations of the earth are more and more observed to have taken place not alone in mountain regions, but even in the regions of plains." Further, it is possible that the outlines of some of the geosynclines traced on this map, a part of which are still hypothetical, would have to be corrected. With this reservation in mind, the following observations can be made.

It may be readily seen that the general sequence of the petroliferous deposits closely follows the path of the synclinal network. About 90 per cent. of the deposits are either included between the borders of the synclinal belts or located in their immediate neighborhood. Approximately 10 per cent. only of the total is to be found at some great distance outside. Such are the deposits of the western coast of Africa, of Southern Australia, of the Red Sea, and some isolated and little known or unimportant deposits in Mongolia, Eastern Siberia (Tchaun Peninsula) and Northwestern Alaska.

The relation of the two last groups (Tchaun Pen. and N. W. Alaska) with the geosynclinal branch of the Northern Pacific is uncertain, as the northern limits of this syncline are still in doubt. The other deposits follow secondary lines of deformation, and many of these belong to this class of lines which Suess has termed "disjunctive"; viz., they are breaks that mark out zones of collapse (*effondrements*). Such are the fractures which have determined the curious depression of Lake Baikal, or the remarkable group of faults that extend from Syria, through the valley of the Jordan, the Dead Sea and the Gulf of Akab, to the cliffs of Abyssinia and hence as far as the neighborhood of Mozambique. These forms are characteristic of disruptive movements having taken place through already consolidated territories, and they seem to imply the break of some fold in the process of formation through grounds that were not plastic enough to allow the completion of the folding. It is remarkable that the hydrocarbon deposits which have been found so far along these secondary lines of distortion are themselves secondary in importance,

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<sup>13</sup> De Launay: *La Science Géologique*, quoted by E. Coste, *Trans.* (1914), 48, 514.

when compared to many of the deposits that stake out the principal geosynclinal belt.

The result of the above is that petroliferous deposits of Mesozoic and post-Mesozoic ages are essentially distributed on the globe along the lines of dislocation characteristic of these ages, and they become thus apparently connected with the contemporaneous earth movements.

There are many reasons to think that the same conclusion would be reached from the comparison between geosynclines and petroliferous deposits of pre-Mesozoic age. Such a view is strengthened, for instance, by the general disposition of the Paleozoic deposits of North America in regard to the Appalachian belt of deformation, from the basin of the St. Lawrence to the Rocky Mountains, or along the borders of the "Canadian Shield," and by other facts. This would imply a general law, that we may tentatively express as follows: *The petroliferous deposits of any given period closely follow the principal areas of deformation of this same period.*

This general, apparent relationship between the areas of distortion and the loci of deposition of the hydrocarbons, or between earth movements and petroliferous accumulations, becomes still more precise with the detailed study of certain oil regions. It is to be expected that the simpler the order of deformation, the more distinct the relations will appear. We are thus brought to consider, at first, regions where the structure is easily interpreted, or at least where it has not been obscured by the superposition of a complex series of stresses, varying in age, direction and intensity. The writer has pointed out such regions as the Eastern Oil Belt of the United States and the belt of the Carpathians in Central Europe, where these conditions are met.<sup>14</sup> In the first instance, the deformation may be interpreted as due to a single dominant thrust; in the second, the thrust would have been repeated, in the same direction, at several distinct periods. The results are represented schematically on Fig. 2 and Fig. 3. Fig. 2 shows the location of the principal oil and gas fields of Pennsylvania and their relation to the Appalachian ridges.<sup>15</sup> Fig. 3 shows the disposition of the principal oil fields of Galicia, and their relation to the trend of the Carpathian range. In the first case, Fig. 2, we have a sequence of simple, broad, and more or less parallel anticlines, merging into monoclines and terraces at the farther end of the distorted area. In the second (Fig. 3), a more complex structure has arisen, due to faults and overthrusts, but it is still anticlinal in its essence and is equally characterized by parallel, concentric lines of

<sup>14</sup> *Bulletin* No. 115 (July, 1916), 1139.

<sup>15</sup> After C. A. Ashburner and J. P. Leslie: *U. S. Geological Survey* (1900-01), 22d *Annual Report*, part 3, 579. The reader is also referred to the general map of the eastern oil and gas fields near the Appalachian Basin, by David B. Reger, in *Bulletin* No. 117 (September, 1916), 1711.



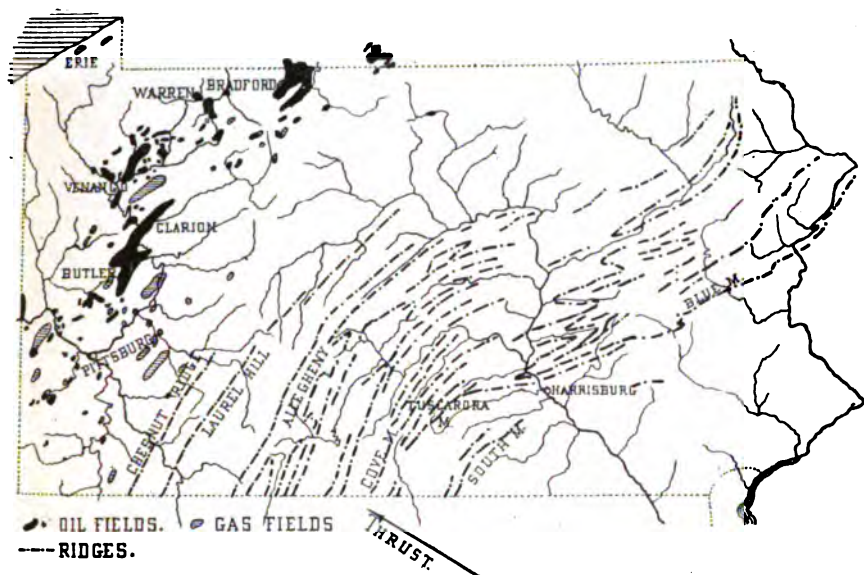


FIG. 2.—DISPOSITION OF THE PRINCIPAL OIL AND GAS FIELDS OF PENNSYLVANIA AND THEIR RELATION TO THE APPALACHIAN RIDGES.

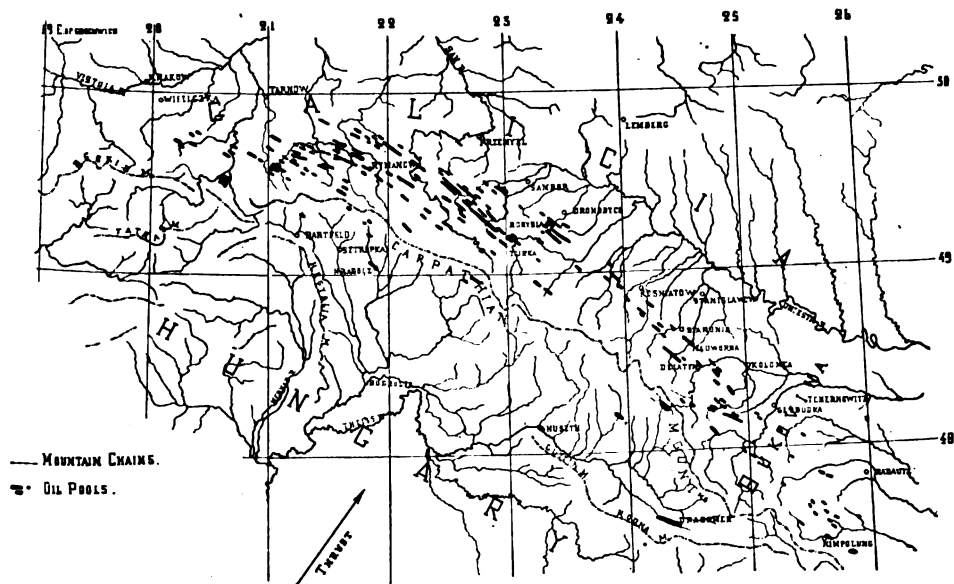


FIG. 3.—DISPOSITION OF THE OIL FIELDS OF GALICIA AND BUKOWINA IN RELATION TO PHYSIOGRAPHIC FEATURES. (AFTER SIR BOVERTON REDWOOD, *Petroleum*, Pl. 2.)

distortion. In either case, the conformity of the trend of the oil deposits with the lines of deformation is impressive, and this unvaried sameness in the effects brings one naturally to infer an identity of causes. If the thrust had, in both instances, at the same time, shoved and folded the strata and concentrated the hydrocarbons along the lines of lesser pressure, the result would not be different. The writer has developed elsewhere the mechanical process through which such a result may have been brought about.<sup>16</sup> It is further to be remarked that a common cause

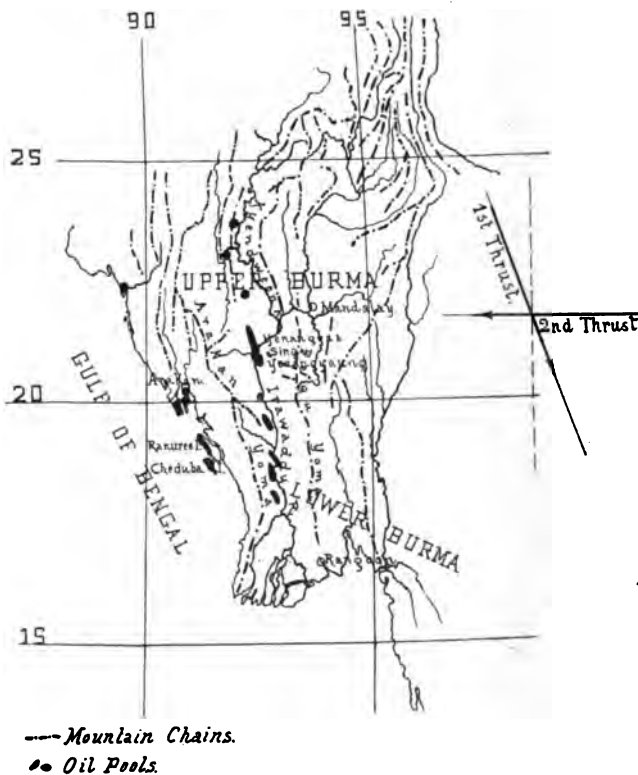


FIG. 4.—OIL FIELDS OF BURMA.

for earth deformation and hydrocarbon accumulation implies the simultaneousness of the two phenomena. Deformation and accumulation would thus be related not only in *space*, but also in *time*. But should this be true, a further conclusion, by no means less important, would follow. If both phenomena, distortion and accumulation, have, so to speak, a common date, then the hydrocarbons must have already existed in the strata when the action of the thrust has taken place, or, in other words, the formation of the hydrocarbons must have anteceded the earth's dis-

<sup>16</sup> *Bulletin* No. 115 (July, 1916), 1139 and No. 125 (May, 1917), 871.

tortion resulting from the thrust, and this formation can in no way be connected with fissures and volcanic phenomena of later date.<sup>17</sup>

Let us now consider a somewhat more complex instance. Fig. 4 is a sketch of the disposition of the Burma oil fields, whose location will be found on the map (Fig. 1) near 92° long. E. of Paris and 20° lat. N. The Burma oil fields form two parallel series. The first extends inland along the Kyendween and Lower Irawaddi valleys, between the Pegu-Yoma mountains on the east and the Arakan-Yoma chain on the west; the second follows the eastern shores of the Gulf of Bengal, on the western

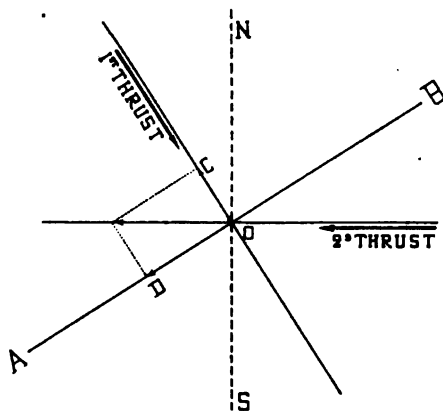


FIG. 5.

side of the Arakan ridge. Both series form a string of isolated deposits, whose course follows the trend of the geosynclinal belt, which runs approximately north-south.

Two flexuring movements have been recognized in this region. An earlier and rather gentle movement caused by a thrust directed from N. 20° W. to S. 20° E., and a later and more important one caused by a thrust directed from east to west. Now, when a series of strata are submitted to two consecutive thrusts coming from two directions inclined at an angle, the first thrust will tend to produce folds normal to its course, like *AB* (Fig. 5). The action of the second thrust is better understood if this force is decomposed into two components: one, *OC*, normal to the fold

<sup>17</sup> This does not imply that during the simultaneous process of distortion and accumulation, or after its completion, some new elements may not have been eventually introduced from below, through crevices or fractures, among the already formed hydrocarbons. This would furnish a key to the presence of helium, for instance, in some natural gas, or may help to explain the peculiar composition of the so-called "connate waters," in other places. But such an action could only be secondary, not primary; local, but not general; and the hydrocarbons, as a whole, could not be genetically related with it. In the same way, it frequently happens that lignites or coals, whose organic origin is undeniable, are locally modified by secondary actions, like dynamometamorphism, or by heat due to the neighborhood of eruptive bodies.

*AB* already produced, and the other, *OD*, parallel to it. The first component, *OC*, will simply tend to exaggerate the fold *AB*; the second, *OD*, will compress this fold along its axis. The final result will depend on the relative intensity of the two forces that act in succession and on the angle at which they meet. If the two forces are nearly at right angle, which would reduce the component *OC* to a minimum, and if the first thrust is a gentle one, giving rise to large undulations of small relief, while the second is much more powerful, the mean strike lines of final deformation will be in agreement with the second movement. But the resulting folds, instead of being continuous anticlines of great length, will become a succession of shortened anticlines or "domes," separated from each other by depressions corresponding, in a general way, to the synclinal areas of the first flexures. This is precisely what seems to have happened in the Burma region, and the oil fields, the Yenangyung, the Yenangyat-Singu and others, are located on the resulting "domes."<sup>18</sup>

An interesting observation has been made in this connection, that throws some further light upon the relations between deformative movements and hydrocarbon accumulations. ". . . Petroleum," writes E. H. Cunningham-Craig<sup>19</sup> "has never been obtained in paying quantity in any field (of the Burma region) that does not show some traces of the earlier movement, even though these traces are often almost obliterated by the much more powerful later movement. It would seem that there has been a preliminary concentration of the petroleum contents of the strata toward the earlier flexures, which concentration has been greatly increased afterward by the later and greater flexuring." This observation tends to make evident that, in this case, accumulation has taken place in two successive and separate stages, each of these being related to a corresponding distinct period of deformation. During the first stage, a first accumulation of Appalachian type would have been formed, by which the hydrocarbons of the surrounding area would have been concentrated away from the thrust and along the lines of lesser pressure, *i.e.*, along anticlinal folds directed W. 20° S. to E. 20° N. With the intervention of the second thrust, the concentration would have been completed toward the resulting "domes," which, by the laws of mechanics, would represent the final areas of minimum pressure. Thus, once more deformation and accumulation are found to be related in space and in time. Not only would accumulation have taken place where deformation occurred, and only there, but the two phenomena would have been synchronous and the first would have been essentially dependent on the second.

To sum up: It has been shown that petroliferous deposits, when they are considered as a whole, are found to be distributed along the principal

<sup>18</sup> E. H. Pascoe: The Oil Fields of Burma. *Geological Survey of India, Memoirs* (1912), 40, 1, Pl. 8, 14, 15 and 19.

<sup>19</sup> *Oil Finding*, 80.

zones of dislocation of the globe, and more, that deposits of a given age follow the trend of the dislocations of this same age. It has been shown also, that in the three particular instances considered (in the Appalachian, the Carpathian and the Burmese regions), the relations noted between the petroliferous deposits and the general structure of the ground suggest a common cause for both the deformation and the hydrocarbon accumulation. And finally, it has been remarked that this identity of cause would require the simultaneousness of the effects produced, or, in other terms, that deformation and accumulation would have to be understood as synchronous and closely allied phenomena.

If now we remember: that the three instances quoted refer to three of the most important oil regions of the world; that these regions are widely separated, in three different continents (America, Europe and Asia); and that the deposits themselves date from various ages (from the Paleozoic to the Tertiary); it becomes evident that the constant relations which have been pointed out cannot be interpreted as local or regional, any more than as transient or exceptional. Their permanent character implies a law of a general order.

There are many oil regions in which these same apparent relations are to be found. But there are also many areas in which the mechanism of distortion has been so complicated as to render its interpretation difficult, at least for the present. So that it would be perhaps too soon to try to formulate this probable law in definite terms. But the facts set forth would at least justify the expression of a working hypothesis, that further investigations may amend or supplement. This is *that petroliferous accumulations are coincident with diastrophic deformations, synchronous with them, and an immediate result of them.*

There are some other remarks that the inspection of the map (Fig. 1) brings forward. The writer wishes to point to one of these: it is the peculiar position occupied by certain oil regions in regard to the trend of the synclinal belt of deformation, such as the region of Borneo, and the area comprising the northeastern coast of Mexico and the Gulf coastal plain region of Texas and Louisiana, farther north. Both regions are located at points where the belt is bent abruptly, and on the inside or concave portion of the curve. Such regions would probably be submitted to peculiar stresses. Besides flexuring by compression, there would be a tendency to deformation by torsion. Daubrée's experiments have shown that torsion may give rise to a system of conjugated faults, equally inclined on the principal axis of torsion. It is significant that if, on one hand, the lines of fractures, resulting in igneous dikes and intrusions, of the Mexican oil fields,<sup>20</sup> and, on the other hand, the faults of the Gulf coastal plain region (whether ascertained faults or hypothetical faults

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<sup>20</sup> Huntley: *Trans.* (1915), 52, 302, Fig. 2; and 310, Fig. 6.

according to the views of Harris)<sup>21</sup> are recorded on the map, they present essentially the appearance of a continuous system of the kind. What may have been the influence of such deformation, combined with deformations due to other causes, on the accumulation of the oil, is still to be seen. But that some kind of relation would exist, is probable in the opinion of the writer. Should this point ever be investigated, it would be prudent to remember that the two groups of conjugated lines (right and left) of Daubrée's experiments are not identical in value. When a

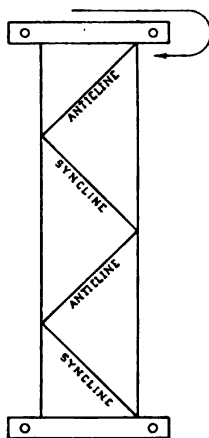


FIG. 6.

flexible sheet, instead of the rigid glass plate used by Daubrée, is submitted to torsion, the motion following the direction of the hands of the clock, the lines of deformation will appear as indicated in Fig. 6. One set is concave (synclinal), the other is convex (anticlinal). If the torsional movement is reversed, the lines will be symmetrically reversed also. Thus, torsion may give rise to flexures; and wherever it produces faults, one set of these would have a tendency to open from above and the other set from below, the maximum relief of stresses being found at the crossing point of the two sets.

<sup>21</sup> G. D. Harris: *U. S. Geological Survey Bulletin* No. 429.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Funnel and Anticlinal-ring Structure Associated with Igneous Intrusions in the Mexican Oil Fields

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(St. Louis Meeting, October, 1917)

### INTRODUCTION

For a number of years the senior author has been interested in the various geologic and engineering problems involved in the development of the petroliferous districts of northeastern Mexico, having in a previous paper<sup>1</sup> endeavored to present a summarized statement of the information relating to the geology and soil resources of this region.

Of the many interesting geologic phenomena, perhaps those which have been the object of more speculation relate to the igneous intrusions which, in places, have a controlling influence on the accumulation of oil in commercial quantities, a relationship which has been tentatively explained by various observers with as many interestingly different views.

The authors of the present paper aim to record, in the course of a general discussion of the various viewpoints, some additional data, and present further tentative conclusions with a view to rounding out an up-to-date summary of the subject.

The greater part of the area in question lies in the State of Vera Cruz, only a northwest fraction being in the State of San Luis Potosi. Vera Cruz is bounded on the north by the State of Tamaulipas, the northernmost of the Mexican States along the Gulf.

The topography of the States of Tamaulipas and Vera Cruz and of Texas and Louisiana to the north, is controlled by the Gulf coastal plain which in northern Vera Cruz has an average width of about 60 miles, the transitional topography between that of the rugged flanks of the Sierra and the lowlands of the coast being made up of a series of terraces and irregular hills and valleys.

### GEOLOGY AND STRUCTURE

The greater part of the mainland of Mexico is made up of Tertiary and later effusive rocks and of sedimentaries of Cretaceous age. The igneous rocks form a belt from 150 to 300 miles wide, spurs of which skirt

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<sup>1</sup> Bibliography, No. 18.

the coast of the northwestern State of Sonora and after receding from the coast continue to the southeast, crossing the central part of the republic in a more easterly trend along the general latitude of Mexico City. This igneous belt, which attains its greatest development in the Sierra Tarahumare in Chihuahua, the Sierra Madre in the States of Durango and Zacatecas, and in the central States to the north of Mexico City, forms the high Mesa Central (central plateau) which has an average elevation between 5,000 and 8,000 ft. (1,524 and 2,438 m.). North and east of this, along the plateau and down the steep decline to the Gulf, extends the larger of the Cretaceous areas which is folded and contorted along the eastern escarpment of the Sierra Madre Oriental and reaches the coast as a gentle low-dipping monocline covered in places with Tertiary and a thin veneer of Quaternary formations. The petroliferous zone is situated on this monocline which forms the coastal plain along the Gulf of Mexico, its geologic features being modified locally by spurs of the belt of effusive rocks to the west.

The formations involved in the geology of the oil region are shown in the accompanying table, the tentative classification adopted in the present paper correlating the formations in the order of their age as follows:

1. *Lower Cretaceous limestones* comprising a series with an approximate aggregate thickness of 10,000 ft. (3,048 m.).
2. *Upper Cretaceous limestones* and shales, having an estimated thickness of 500 ft.<sup>2</sup> (152.4 m.).
3. *Cretaceous-Eocene shales* about 3,000 ft. (914 m.) thick.<sup>3</sup>
4. *Later Tertiary limestones, sands and clays*, about 2,000 ft. (609 m.) in thickness, of Oligocene and later age.
5. *Igneous intrusions* of late Tertiary or possibly early Quaternary.
6. A thin cover of Quaternary and recent deposits.

A general idea of the areal distribution of the geologic formations and the location of the principal igneous intrusions is given in a map by Geoffrey Jeffreys recently published.<sup>4</sup>

## THE IGNEOUS INTRUSIONS

### *General Statement*

One of the characteristic features of the Mexican oil fields is the great number of igneous masses which dot the region sometimes in the form of

<sup>2</sup> The existence of an unconformity at the base of the San Felipe series, which was hinted at in previous papers has been further confirmed by the observations of C. W. Washburne. Unconformities very likely exist also at the bases of the Mendes and San Fernando series.

<sup>3</sup> Later investigation of the fossil fauna found in the sandstones at the top of the Cretaceous-Eocene shales in the vicinity of Alazan shows that they bear a very marked relation to the Eocene of California with which they had been tentatively correlated.

<sup>4</sup> Bibliography, No. 21, p. 302.



conical peaks, breaking the monotony of the level plain. The greater number of these cones and associated dikes occur in the southern part, attaining their greatest development in the Otontepec range which covers an approximate area of 35 square miles (90.65 km.<sup>2</sup>) and attains a height of about 3,500 ft. (1,067 m.), decreasing in frequency towards the east and north as the distance from the Sierra Madre increases and disappearing entirely before the Rio Grande is reached.

According to De Golyer,<sup>5</sup> the area of greatest igneous activity, roughly triangular in outline, is defined by a north-south line through Furberro, second line through Tantoyuca and Ozuluama and a third line along the front of the Sierra Madre Oriental. Within this area are located the productive fields of Casiano, Potrero del Llano, Dos Bocas, Alamo, Los

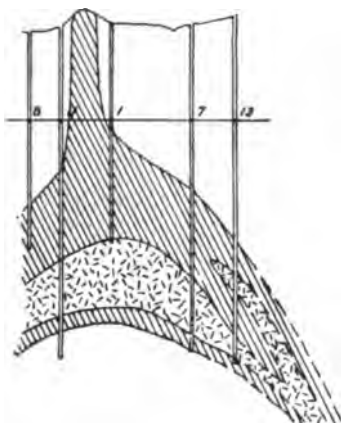


FIG. 1.—SECTION ACROSS A PORTION OF THE FURBERRO SILL, SHOWING UNDERGROUND STRUCTURE. THE CROSS-HATCHED AREA ABOVE AND BELOW THE BASALT SILL INDICATES THE ZONE OF METAMORPHOSED SHALE. (AFTER DE GOLVER.)

Naranjos and others, while the most important districts so far discovered outside its boundaries are Ebano, Topila and Panuco in the northern region between the Tamesi and Panuco Rivers.

Most of the surface indications of petroleum are closely associated with the exposed intrusions, such being the case at La Pez, Chijol, San Geronimo, La Merced, Rancho Abajo, Monte Alto, Los Higueros, Dos Bocas, Casiano, Cervantes, Tres Hermanos, Tinaja, Ojo de Brea, Chapopotillo, Monte Grande, Moralillo, Cerro Azul, Juan Felipe, Las Borrachas, Piedra Labrada, Cerro Viejo, Potrero del Llano, Tierra Amarilla and Chapopote Nunez. As it is evident that not all of the intrusions reached the surface, some of the surface "showings" of petroleum near which no intrusive masses are visible may be closely associated with the underlying igneous masses.

<sup>5</sup> Bibliography, No. 23, p. 658.

An intermediate step between the exposed and buried intrusions may be represented by outcropping masses of metamorphosed shale, such as the sharp twin peaks of Furbero (see Fig. 1), the main elevation at Cerro Azul, the Cerro de Zaragosa cone and the peak at Taninul, all of which resemble basaltic cones in outline and likely are representative surface phenomena of the underlying intrusions. Huntley<sup>6</sup> in discussing these topographic features and their association with hidden intrusions cites as an example Cerro Taninul of which he says:

"The western one is composed of basalt, while the smaller eastern peak (Taninul) shows only large blocks of a metamorphosed marl containing numerous small siliceous nodules. This peculiar rock is found at a number of places in the vicinity of large intrusions of basalt and has been caused by the circulation of underground currents, probably both hot and more or less mineralized, as pyrite is frequently found nearby."

The senior author has observed a similar phenomenon along the base of Cerro Azul.

### *Form of Intrusions*

It is interesting to record in this connection the different views advanced by various geologists regarding the form of the intrusions: Villarello<sup>7</sup> believes that some of the hills, like that of La Pez, are in reality small cones or craters through which took place the flow of basaltic tuffs which extend a little over the plain; Ordonez<sup>8</sup> and Coste<sup>9</sup> hold the view that the volcanic disturbances were mostly of the explosive type drilling upward through the sedimentaries and opening narrow pipes, these explanations conforming in part with the views of Harker,<sup>10</sup> who says that small masses of molten material, by a process of "overhead-stoping," are able to form almost cylindrical pipes through the intruded rocks.

F. G. Clapp and I. C. White<sup>11</sup> ascribed to the intrusion the form of a more or less irregular cone in a normal position, the vertex of which may or may not reach the surface. This theory, however, is no longer tenable in view of the results of actual experience, as many of the producing wells like those of Ebano are located within "stone's throw" of exposed intrusive plugs and in other cases the basalt has been penetrated for several feet and the well continued through sedimentaries to the oil reservoir below, conditions which could not obtain were the "normal cone" theory correct. In order to account for this discrepancy, the senior author performed in 1912 a rather crude experiment<sup>12</sup> from which one of the tentative conclusions drawn was that the thickness of the basalt

<sup>6</sup> Bibliography, No. 21, p. 312.

<sup>7</sup> Bibliography, No. 6, pp. 39-40.

<sup>8</sup> Bibliography, No. 10, p. 1020.

<sup>9</sup> Bibliography, No. 15, p. 504.

<sup>10</sup> Bibliography, No. 8, p. 86.

<sup>11</sup> Bibliography, No. 12, pp. 277-278.

<sup>12</sup> Bibliography, No. 13.

intrusions increases rather than decreases from its deep-seated source to the surface, thus forming an inverted irregular cone, or more nearly a "mushroomlike" plug with projecting irregular annular tongues along the bedding planes of the disturbed intruded beds.

Sills and other forms of intrusions which do not outcrop, like the underlying sill at Furbero, may be formed as suggested by De Golyer,<sup>13</sup> by an igneous mass ascending in the form of a dike which on encountering a steeply dipping bedding plane of little resistance, is diverted into that plane as a sill or sheet. The statement of Dumble<sup>14</sup> that the basaltic material in the producing fields occurs mostly in the form of sills or intercalated sheets of moderate thickness would indicate that this is a prevalent rather than an exceptional occurrence.

Although some of the earlier investigators<sup>15</sup> claimed that the occurrence of dikes was practically unknown in the Mexican fields, later work disclosed the fact that dikes are quite common in this region; Jeffreys, who made a thorough study of them, placing considerable stress upon their importance as factors in the accumulation of oil.

### *Petrology of Intrusions*

The material which makes up the intrusions has been generally classified as basalt accompanied, as at La Pez and La Dicha, by volcanic ash and agglomerate.

Ordóñez<sup>16</sup> reports some of the pipes made up of basaltic lava, sometimes olivine basalt and others basaltic tuff; both Clapp<sup>17</sup> and Coste<sup>18</sup> say that the volcanic necks are composed chiefly of olivine basalt, while De Golyer<sup>19</sup> who has observed the occurrences in the southern districts says that:

"Many of the plugs, such as Cerros Palma Real and Pelon, consist altogether of volcanic breccia composed in part of sedimentary rocks, while others consist of a relatively small chimney of basalt surrounded by a rather wide aureole of brecciated material."

The breccia which is found associated with intrusions is composed largely of fragments of limestones and shales carried in suspension in the semi-viscous igneous mass, metamorphosed and cemented upon the solidification of the molten material. It seems very probable that many of the hills, which are composed in most part of such brecciated rocks under

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<sup>13</sup> Bibliography, No. 20, p. 278.

<sup>14</sup> Bibliography, No. 19, p. 254.

<sup>15</sup> Bibliography, No. 6, p. 40 and Bibliography, No. 17, p. 860.

<sup>16</sup> Bibliography, No. 5, p. 247.

<sup>17</sup> Bibliography, No. 12, p. 379.

<sup>18</sup> Bibliography, No. 15, p. 510.

<sup>19</sup> Bibliography, No. 23, p. 659.

varying degrees of alteration, may give place deeper down to the igneous mass which has caused the brecciation and cementation of the fragmentary material.

A microscopic study of the igneous material has been made by Huntley,<sup>20</sup> who published some photomicrographs of thin sections of basalt collected from dikes in the central districts; he reports the presence of labradorite, olivine, augite and magnetite in a rock which he classes as olivine dolerite.

It will seem, therefore, that lacking a more thorough petrographic study, the intrusive rocks may be classed as belonging to the basalt group consisting for the most part of basalts, dolerites and diabase sometimes with vesicular structure and accompanied by breccia, volcanic ash and agglomerate.

#### *Relationship of Intrusions to Regional Structure*

The period of vulcanism in this region was subsequent to that of folding, and the areal distribution of dikes and plugs indicates that the molten material in seeking a relief of pressure arose to the surface, or to higher planes of equalized pressure within the sedimentaries, mainly along fissures or planes of weakness in the overlying beds, the general arrangement of the dikes and the existence of isolated plugs apparently along definite lines suggesting the existence of well-defined fractures; Jeffreys noted that although the intrusive masses occur throughout the coastal plain, there seems to be a very pronounced zone of vulcanism running northeast-southwest, passing from Chicontepec to Tantima, Chapopote and thence to Dos Bocas.

Huntley<sup>21</sup> ascribes the origin of these fractures and planes of weaknesses to the main intrusion of Sierra Otontepec, and the location of the radiating and tangential dikes in the immediate neighborhood, he claims, may be due to secondary fracturing caused by this large intrusion and its subsequent shrinking. De Golyer<sup>22</sup> advances the explanation that the intrusions took place through faults and fissures developed in the rocks by shearing stresses and strains resulting from conflict in the forces which folded the rocks originally.

It may be noted that the plugs whose location is controlled by structural lines of weaknesses present some analogy to the saline domes in the oil fields of the Texas Louisiana coastal plain.

#### *Localized Effect of the Intrusions*

The intruding molten material produced a certain amount of metamorphism in the surrounding sedimentaries, both as a direct effect of its

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<sup>20</sup> Bibliography, No. 21, p. 281.

<sup>21</sup> Bibliography, No. 21, p. 281.

<sup>22</sup> Bibliography, No. 23, p. 658.

high temperature and as a result of the hydrothermal alteration caused by the circulation of heated underground waters. The metamorphic action on the shales accompanied by an increase in density and diminution of volume gave rise to a considerable amount of brecciation upon the consolidation and contraction of the molten mass, and this resulted in the formation of an aureole of material more porous than the surrounding unaltered beds.

The metamorphic action on the limestones, in the immediate vicinity of the intrusions very likely caused a decrease in volume and the formation of cavities as a result of dolomitization, and the igneous rock itself was probably altered by the hot circulating waters, with the probable increase of the void space; a specimen of the basalt which makes up the Furbero sill showing, according to De Golyer and Norman,<sup>23</sup> a porosity of 6 per cent.

The localized effect of the intrusions on the structure of the surrounding sedimentaries presents some uniform features with minor differentia-



FIG. 2.—SECTION ACROSS THE BINN OF BURNTISLAND, FIRTH OF FORTH, SCOTLAND.

tion according to the preëxisting structure, the form of the intrusive mass and its position above or below ground.

It is reasonable to suppose that the intrusions that do not reach the surface, such as the sill that controls the accumulation of oil in the Furbero field (see Fig. 2), irrespective of their form and preëxisting structure, have produced, as a rule, a certain amount of upward folding of the overlying beds, thus initiating or accentuating a domical structure. It should be noted, however, that minor features of the resulting structure will be controlled to some extent by the attitude of the intruded beds before the intrusion took place.

The structural features developed by basalt dikes, pipes and necks which reach the surface were studied by Heaphy<sup>24</sup> as early as 1860 and further corroborated by Geikie, 20 years later,<sup>25</sup> these investigators observing that the beds near basalt intrusions dip rather steeply toward the intrusions (see Figs. 2 and 3), a phenomenon which later has been observed in the Mexican oil region by Jeffreys, De Golyer,<sup>26</sup> Stewart<sup>27</sup> and others.

<sup>23</sup> Bibliography, No. 20, p. 268.

<sup>24</sup> Bibliography, No. 1, p. 244.

<sup>25</sup> Bibliography, No. 3, pp. 468-470.

<sup>26</sup> Bibliography, No. 23, p. 659.

<sup>27</sup> Bibliography, No. 22, p. 11.

The inclination of the strata toward the intrusions is accounted for by the settling or "drag-back" of the igneous material while cooling, due to its contraction upon solidification; a phenomenon which also explains the extensive amount of brecciation in the indurated shales surrounding the plugs. The amount of shrinkage of molten magmas under these conditions has been estimated to reach as high as 10 or 11 per cent. Barus<sup>28</sup> found that diabase undergoes the following contraction upon cooling from 1,500° C. to 0° C.: from 1,500° C. to 1,093° C., 3 per cent.; upon solidification at 1,093° C., 3.55 to 4 per cent.; from 1,093° to 0° C., 4 per cent. This phenomenon would readily account for the downward drag of the plug with the attendant brecciation and "funnelling" of the surrounding sediments.

In order to allow the necessary space for the invading magma or breccia, and irrespective of the preëxisting structure, a certain amount of upward folding had to take place while the intrusion was in progress,



FIG. 3.—SECTION ACROSS LARGO LAW, SCOTLAND. B, BASALT; t, VOLCANIC TUFF; C, CARBONIFEROUS STRATA. (AFTER GEIKIE.)

followed during the cooling and settling stage by the dragging down of the shattered edges of the displaced beds, to an inclined or even vertical position in the immediate vicinity of the intrusion, thus giving rise to an outer annular anticlinal fold and an inner concentric funnel depression immediately surrounding the igneous mass, the anticlinal-ring-and-funnel structure thus developed being made up of a series of minor complex asymmetric folds.

#### *Observations on Similar Phenomena in Other Parts of the World*

The phenomena of the inward dip of beds surrounding a volcanic neck is by no means uncommon, the earliest observations, as mentioned before, being recorded by Heaphy<sup>29</sup> in a description of the volcanic country of Auckland, New Zealand. At a later date, Geikie cites<sup>30</sup> the occurrence in Central Scotland, especially in the basin of the Firth of Forth, of volcanic necks, dikes and sheets which resemble very much those of the Mexican fields in topographic expression, composition and structural

<sup>28</sup> Bibliography, No. 4, pp. 36-39.

<sup>29</sup> Bibliography, No. 1, p. 244.

<sup>30</sup> Bibliography, No. 3, pp. 459-461.

relations. Here also the material filling up the vents consists of molten rocks which have flowed through the neck and solidified, and breccia or volcanic detritus made up of fragments of the intruded strata, blocks of the intrusive material itself and tuffaceous material, the whole firmly cemented by either the intrusive lava or secondary alteration products, the igneous material making up the intrusions being usually basalt, dolerite or porphyrite.

In discussing the structure of the beds adjacent to these intrusive pipes, Geikie writes:<sup>21</sup>

"In the majority of the shore-sections, a remarkable change of dip is observable among the strata round the edge of each vent. No matter what may be the normal dip of the locality, *the beds are bent sharply down toward the wall of the neck*, and frequently placed on end. . . . This is precisely the reverse of what might have been anticipated, and can hardly be due to the upward volcanic explosions. It is usually associated with considerable metamorphism of the disturbed strata.

"Taking it in connection with the metamorphism, I am inclined to believe that it took place after the long-continued volcanic action which had hardened the rocks round the volcanic pipe, had ceased, and as a result of some kind of subsidence within the vent. So general is this evidence of downward movement in all the volcanic districts where the necks have been adequately exposed that *this structure may be suspected to be normal to all old volcanic vents.*"

Du Toit<sup>22</sup> observed a similar structure in close proximity to the kimberlite dikes and necks in the Kimberley diamond fields of South Africa and analogous occurrences are reported, by Dr. J. C. Branner, from Panama, where numerous basalt necks, some of which send out branches into the surrounding beds, are found cutting the sedimentaries along the Canal. Concerning the shape of the intrusion which forms Gold Hill, General Goethals writes:<sup>23</sup>

"Gold Hill is of basalt, thrown up in a molten state through the sedimentary deposits that already existed, and poured over the deposits on either side of the stem, giving to the vertical section, the general shape of a mushroom."

## THE INTRUSIONS AND THE OIL FIELDS

### *Relation of the Intrusions to the Origin of Petroleum*

The association of the Mexican oil fields with igneous activity has been advanced by Coste as an argument in favor of the inorganic origin of petroleum; thus, in discussing the relation between the occurrence of oil and igneous intrusions, he writes:<sup>24</sup>

". . . it is manifest that these Mexican petroleum deposits are directly connected with vulcanism, and due to solfataric volcanic emanations accompanying the upheaval of the basaltic cones."

<sup>21</sup> Bibliography, No. 3, pp. 469-470.

<sup>22</sup> Bibliography, No. 9, pp. 114-115.

<sup>23</sup> Bibliography, No. 27, p. 30.

<sup>24</sup> Bibliography, No. 15, p. 504.

The geologists who advocate the organic origin of the Mexican petroleum believe that the source of the oil may be ascribed to the original organic matter in what is now the Tamasopo limestone<sup>35</sup> or that entombed in the overlying shales, leaving to the direct action of the heat of the molten intrusions the secondary role of partially distilling the comparatively small amount of organic material which came within the influences of the heated masses.<sup>36</sup>

*Effect of the Intrusions on the Migration and Accumulation of Oil*

Although the relation between the origin of petroleum and the intrusions is a matter of conjecture, the influence of the intrusions in the migration and accumulation of the oil should not be underestimated. In the first place, the contact zone of metamorphosed and brecciated sedimentaries with their extensive fractures provides a system of channels for the migration of the oil in the Tamasopo to storage reservoirs in higher planes, or to the surface if the channels break the exposed beds. Furthermore, the brecciated zones, made up partly of well-like channels, when effectively capped, would form an excellent container for commercial deposits of petroleum, having a large vertical drainage area of connected passages through which the gas, oil and underlying water could flow undisturbed by the crumbling and caving attendant upon the flow of these fluids through unconsolidated sediments. The same conditions as regards structure would obtain if the source of the oil were the organic material in the shales above the Tamasopo. In either case, the resulting funnel-and-anticlinal-ring structure would form ideal zones for the accumulation of oil and even if the amount of hydrocarbons disseminated through the sedimentaries were small, the accumulation near the intrusions would be comparatively large owing to their barrier-like action and to the large horizontal and vertical drainage areas contributing to such concentration zones.

The asymmetric anticlinal-ring structure may be considered especially favorable for the accumulation of oil while the central funnels, which might become flooded with oil during the early life of the well, would later show the synclinal tendencies for collecting water, as is the case of Gold Hill in the Panama Canal and in several basalt plugs in Mexico.

Dikes would probably give rise to similar structures rather than to plugs or pipes, but of a more elongated character; of added importance for the accumulation of oil being in this case the effect of dikes in blocking the underground flow of oil and, in many cases, of excluding water from an adjoining producing oil reservoir. Huntley opposes this view, saying that:<sup>37</sup>

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<sup>35</sup> Bibliography, No. 18, pp. 202-206.

<sup>36</sup> Bibliography, No. 25, p. 727.

<sup>37</sup> Bibliography, No. 21, p. 281.



"Basalt dikes cannot be looked upon as barriers to the movement of oil, except in so far as they accompany fractures which tend to trap it."

Jeffreys, on the other hand, lays great stress upon the importance of dikes in governing the migration of oil and water, and finds that these barriers dam the water so effectively that although one well may have "gone to water" its neighbor separated from it by a dike will continue to yield "dry" oil and be unaffected by the adjacent flooded area.

The basalt itself and the metamorphosed zone of shales, as at Furbero, surrounding the vesicular basalt intruded along the bedding planes, because of its altered and brecciated, hence porous, nature, forms likewise reservoirs favorable for the accumulation of commercial quantities of oil. De Golyer and Norman say that:<sup>28</sup>

"... the intrusion would have been as effective in any form which it might have assumed, provided that it did not outcrop;"

and further, that the dominant factor in the accumulation of oil in this instance is the greatly varying porosity of the rock rather than the structure.

#### SUMMARY

As subjects for further study and discussion, the writers present the following tentative conclusions:

1. In a general way, the intrusions which are exposed at the surface occur as mushrooms, pipes or necks, and as dikes, often in a near-vertical position; while those which do not outcrop take more readily the form of sills or minor laccoliths intruded along bedding planes or other planes of weaknesses.

2. The material making up the intrusions belongs to the basalt group of igneous rocks and consists for the most part of basalts, dolerites, and diabase, vesicular in places, and associated with breccia, volcanic ash and agglomerate.

3. Although the location of some intrusions apparently bears no relation to existing structure, the greater number occur along fractures or other lines of weaknesses in the sedimentaries, probably caused by either the forces that produced the folding or those which accompanied the intrusions of the larger igneous masses.

4. The effects of the intrusions on the adjacent strata may be grouped under three heads: first, a metamorphic change of the sedimentaries which involved a decrease in volume with a corresponding increase in porosity; second, the formation of a more or less vertical system of connected channels around the intrusions for collecting the hydrocarbons disseminated in the intruded sedimentaries; third, the development of a peculiar funnel-and-anticlinal-ring structure in the sedimentaries as a

<sup>28</sup> Bibliography, No. 20, p. 268.

result of the uplifting and subsequent drag back by the solidifying igneous mass.

5. The funnel-and-anticlinal-ring structure is by no means uncommon in connection with volcanic necks and dikes elsewhere. Geikie says that this structure may be suspected to be normal to all old volcanic vents.

6. The commercial importance of the technical study of the intrusions and related phenomena lies in the correlation of the character and attitude of the actual oil productive zones with the pre-existing structure and the form and position of the intrusion. As an illustration of the probable effect of the intrusions on the concentration of oil, one may take the case of a plug or dike intruding beds that originally formed an undisturbed monocline (see Fig. 4). It seems reasonable to conclude that the resulting structure would be of the nature of a terrace on the "upper" side of the monocline and a decided asymmetric anticline on the opposite or "lower" side with gradations around the intrusion from one to the other type of fold. Were these facts ascertained, in a particular pool and knowing that the conditions for larger accumulations of oil were more favorable on the "anticlinal" side of the plug, it is obvious that the drilling programs could be more intelligently planned.

7. Although there is no question of the existence of these conditions in the Mexican fields, the writers do not know how numerous such occurrences are and whether other factors are of greater importance in the accumulation of oil in the region, than the igneous intrusions here discussed.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Hancock Jig in the Concentration of Lead Ores.

BY HAROLD RABLING,\* B. M. E., BONNE TERRE, MO.

(St. Louis Meeting, October, 1917)

THE following notes are taken from results obtained on a standard 25-ft. Hancock jig<sup>1</sup> tested during regular operation in the Bonne Terre mill of the St. Joseph Lead Co. The object of the tests was to determine the conditions for most effective work, and the nature of products that could be made.

The material treated was sized between 9-mm. and 2-mm. round-hole screens, and was constant throughout the experiments. Three products were made, concentrates from the first three hutches, middlings from the fourth and fifth, and tailing from the sixth. The middlings were crushed in one pair of Allis-Chalmers Style B 30 by 14-in. rolls, and elevated with the original ore to the 2-mm. screens, oversize of which formed the jig feed.

While this arrangement was very simple, it gave rise to undesirable crowding of the circuit. Table 1 shows tonnage and assay values of the jig feed and products at the beginning of the experiments, and brings out the excessively large amount of middlings, comprising 63.3 per cent. of the total material fed to the jig. The rolls, being heavily overloaded, did very little crushing, a screen analysis of the product showing that only 27 per cent., or 128 tons, was crushed fine enough to pass to the tables. The remaining 347 tons returned to the jig with 400 tons of original ore. At times, this overcrowding became much worse, and the tonnage in circuit grew to enormous proportions. It became imperative

TABLE 1.—*Work of Hancock Jig at Beginning of Tests*

| Material          | Tons per 24 Hr. | Assay, Per-Cent.<br>Lead | Tons Lead per 24<br>Hr. |
|-------------------|-----------------|--------------------------|-------------------------|
| Feed.....         | 750.0           | 4.06                     | 30.500                  |
| Concentrates..... | 16.5            | 70.00                    | 11.550                  |
| Middlings.....    | 475.0           | 3.50                     | 16.625                  |
| Tailing.....      | 258.5           | 0.90                     | 2.325                   |

\* Testing Engineer, St. Joseph Lead Co.

<sup>1</sup> For description of Hancock Jig see *Trans.* (1913), 46, 213.

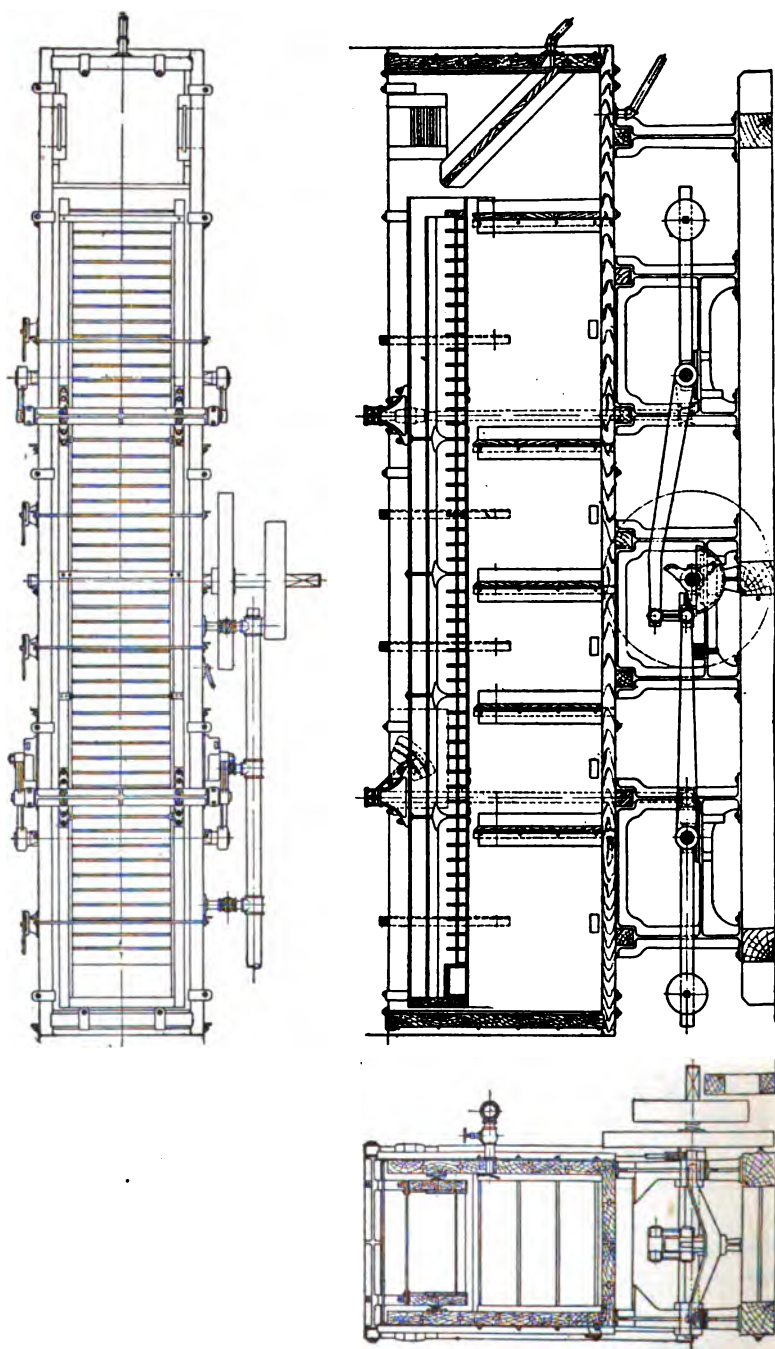


FIG. 1.—THE HANCOCK JIG.

either to decrease the amount of middlings, or to provide more crushing capacity, and series of experiments were begun along both lines. This paper deals only with the attempt to improve the work of the jig as then equipped.

Before commencing the experiments, a combined sizing and sorting test was made on an average sample of the jig feed. Tyler standard screens were used for the sizing test and all sorting was done by hand. Table 2 shows the result of this test, and Figs. 2 and 3 represent the same thing graphically. The horizontal divisions in the diagrams represent the percentages of material on each size, and the vertical divisions the percentages of free galena, middling, and free gangue as obtained by

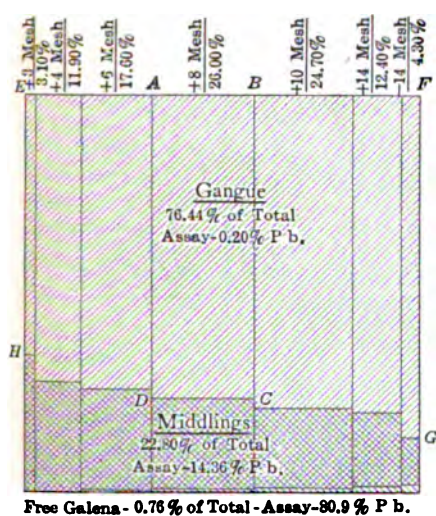


FIG. 2.—SCREEN ANALYSIS OF JIG FEED AND PERCENTAGES OF FREE GALENA, MIDDINGS AND FREE GANGUE IN EACH SIZE.

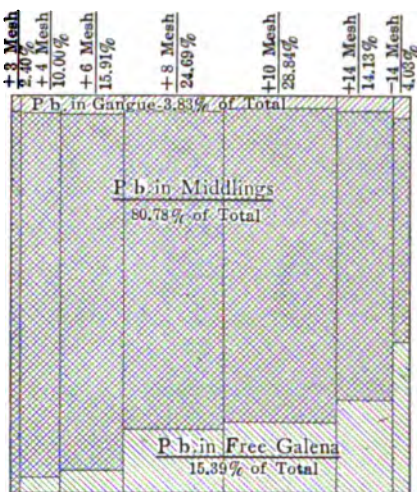


FIG. 3.—SCREEN ANALYSIS OF LEAD IN JIG FEED AND PERCENTAGES OF TOTAL LEAD IN EACH SIZE AS FREE OR INCLUDED GALENA.

sorting. Areas, therefore, represent percentages of the total weight of sample. For instance, in Fig. 2 the length  $AB$  is 26 per cent. of the total length of the diagram, and represents the amount of material between 6 and 8-mesh. The length  $AD$  is 76.65 per cent. of the total height of the diagram and represents the percentage of the 6 to 8-mesh material, which occurs as free gangue. The area  $ABCD$ , therefore, shows the percentage of the total sample occurring as free gangue between 6 and 8-mesh and is equal to 19.927 per cent. of the total area of the diagram, while the area  $EFGH$  shows the total amount of free gangue in the sample, or 76.44 per cent.

The results obtained by this test led to the belief that it was possible to greatly reduce the tonnage of hatched product without affecting the

TABLE 2.—Sizing and Sorting Test on Jig Feed

| Mesh            | Per Cent. Weight | Assay, Per Cent. Lead | Per Cent. of Total Lead Content | Per Cent. of Weight on Each Mesh |       |             | Per Cent. of Total Weight |        |             | Assay, Per Cent. Lead |       |             | Per Cent. of Lead on Each Mesh |       |             | Per Cent. of Total Lead Content |        |             |
|-----------------|------------------|-----------------------|---------------------------------|----------------------------------|-------|-------------|---------------------------|--------|-------------|-----------------------|-------|-------------|--------------------------------|-------|-------------|---------------------------------|--------|-------------|
|                 |                  |                       |                                 | Free Galena                      | Midd. | Free Gangue | Free Galena               | Midd.  | Free Gangue | Free Galena           | Midd. | Free Gangue | Free Galena                    | Midd. | Free Gangue | Free Galena                     | Midd.  | Free Gangue |
| On 3...         | 3.1              | 3.14                  | 2.40                            | .....                            | 34.75 | 55.25       | .....                     | 1.078  | 2.022       | .....                 | 8.64  | 0.22        | .....                          | 95.38 | 4.62        | .....                           | 2.290  | 0.111       |
| On 4...         | 11.9             | 3.41                  | 10.00                           | 0.16                             | 28.00 | 71.84       | 0.019                     | 3.330  | 8.551       | 86.8                  | 11.22 | 0.18        | 4.06                           | 92.14 | 3.80        | 0.407                           | 9.213  | 0.380       |
| On 6...         | 17.6             | 3.66                  | 15.91                           | 0.26                             | 25.86 | 73.88       | 0.046                     | 4.550  | 13.004      | 86.7                  | 12.62 | 0.24        | 6.14                           | 89.02 | 4.84        | 0.976                           | 14.165 | 0.769       |
| On 8...         | 26.0             | 3.85                  | 24.69                           | 0.78                             | 22.57 | 76.65       | 0.203                     | 5.870  | 19.927      | 80.3                  | 13.71 | 0.17        | 16.34                          | 80.27 | 3.39        | 4.080                           | 19.819 | 0.836       |
| On 10...        | 24.7             | 4.73                  | 28.84                           | 1.02                             | 20.69 | 78.29       | 0.252                     | 5.110  | 19.338      | 82.0                  | 18.09 | 0.20        | 17.80                          | 78.90 | 3.30        | 5.140                           | 22.745 | 0.955       |
| On 14...        | 12.4             | 4.62                  | 14.13                           | 1.32                             | 18.81 | 79.87       | 0.164                     | 2.335  | 9.901       | 81.6                  | 17.84 | 0.22        | 23.40                          | 72.79 | 3.81        | 3.300                           | 10.295 | 0.538       |
| Through 14..... | 4.3              | 3.80                  | 4.03                            | 1.80                             | 12.20 | 86.00       | 0.077                     | 0.525  | 3.698       | 80.5                  | 17.45 | 0.26        | 38.15                          | 56.00 | 5.85        | 1.537                           | 2.257  | 0.237       |
| Total....       | 100.0            | 4.06                  | 100.00                          | 0.76                             | 22.80 | 76.44       | 0.761                     | 22.798 | 76.441      | 80.9                  | 14.36 | 0.20        | 15.39                          | 80.78 | 3.83        | 15.380                          | 80.784 | 3.826       |



grade of tailings. Table 3, worked out from the results of the sorting, shows the work that might be expected from a theoretically perfect concentrating machine, and differs considerably from the record of the actual work obtained (Table 1). An unnecessarily large amount of free gangue seemed to be passing through the jig screens, and a sorting test on the jig products (Table 4) confirmed this, the fifth hutch especially containing almost as much free gangue as the tailing.

In the investigations following the above work, special attention was given to the possibility of decreasing the amount of gangue being hatched with the middlings, and a summary of results obtained and conclusions arrived at, is given here.

TABLE 3.—*Theoretically Perfect Work Obtainable on Material Fed to Jig*

| Material          | Tons per 24 Hr. | Assay, Per Cent. Lead | Tons Lead per 24 Hr. |
|-------------------|-----------------|-----------------------|----------------------|
| Feed.....         | 750.00          | 4.06                  | 30.500               |
| Concentrates..... | 5.70            | 80.90                 | 4.753                |
| Middlings.....    | 171.00          | 14.36                 | 24.600               |
| Tailings.....     | 573.30          | 0.20                  | 1.147                |

TABLE 4.—*Summary of Sorting Test on Jig Products*

| Material         | Assay, Per Cent. Pb | Per Cent. of Total Weight |       |             | Assay, Per Cent. Lead |       |             | Per Cent. of Pb Cont. |       |             |
|------------------|---------------------|---------------------------|-------|-------------|-----------------------|-------|-------------|-----------------------|-------|-------------|
|                  |                     | Free Galena               | Midd. | Free Gangue | Free Galena           | Midd. | Free Gangue | Free Galena           | Midd. | Free Gangue |
| Concentrates.... | 74.40               | 64.4                      | 35.6  | .....       | 82.3                  | 60.40 | .....       | 71.1                  | 28.9  | .....       |
| 4th hutch.....   | 6.78                | .....                     | 31.2  | 68.8        | .....                 | 20.70 | 0.48        | .....                 | 95.4  | 4.6         |
| 5th hutch.....   | 1.53                | .....                     | 16.7  | 83.3        | .....                 | 8.33  | 0.17        | .....                 | 90.3  | 9.7         |
| Tailing.....     | 0.88                | .....                     | 15.5  | 84.5        | .....                 | 4.91  | 0.14        | .....                 | 84.4  | 15.6        |

### *Operation of the Jig*

The following factors may be taken as having most effect on the work of the Hancock jig:

1. The speed of the jig.
2. Length of vertical stroke.
3. Length of horizontal stroke.
4. Depth of bed.
5. The size of the grains forming the bed.
6. The specific gravity of the grains forming the bed.

1. *The speed of the jig* was varied between wide limits—from 170 to 210 strokes per minute. The high speeds gave increased tonnage of hutch product, and were objectionable from a mechanical standpoint. At the slow speed of 170, difficulty was experienced in keeping the bed moving freely, it having a tendency to pack, and lose its quicksand effect.

A speed of 190 to 195 strokes per minute was finally decided on as being the most desirable.

2. *The length of vertical stroke*, by affecting the pulsion and suction velocities affects the tonnage of hatched product, increased length causing more material to be hatched, and *vice versa*. It varies between  $\frac{3}{8}$  and  $\frac{3}{4}$  in., the adjustment being made by the operator according to the lead content of the material treated.

3. *The horizontal stroke* should be as long as possible, in order to have, above the bedding, a thin layer of ore. In treating large tonnages, the long stroke is particularly desirable, the improved separation in a thin layer more than compensating for the shorter time the material is under treatment. A minimum horizontal stroke of  $\frac{3}{4}$  in. is used at Bonne Terre, and it is probable that greater capacity could be had by making the stroke much longer.

4. *The depth of bed on the Hancock jig* is determined by the height of the slats above the screen. No experiments were made with variations of this distance, all jigs being operated with 3 in. of bed depth.

5 and 6. *The size and specific gravity* of the grains forming the bed are determined entirely by the size of the jig screens. No artificial bedding has been supplied at any time, as experiments have shown that with a carefully designed system of screens any desired bed can be produced and held, relying on the ore to supply all the material necessary. The nature of the bed being the most important factor influencing the work of the jig, the main part of the investigation was taken up by the effort to work out a system of screens that would give satisfactory work under average conditions, and would be flexible enough to take care of extreme variations in the grade and nature of the material treated.

The system that was finally evolved is given in Table 5, and while it would probably only be applicable to the Bonne Terre ore, the principles of its evolution may be found useful in the solution of similar problems elsewhere.

TABLE 5.—*System of Screens Used on Hancock Jig*

|                         |     |         |
|-------------------------|-----|---------|
| 4-mm. round hole to rib | 3   | 3 ribs  |
| 5-mm. round hole to rib | 15  | 12 ribs |
| 7-mm. round hole to rib | 16  | 1 rib   |
| 6-mm. round hole to rib | 22  | 6 ribs  |
| 7-mm. round hole to rib | 30  | 8 ribs  |
| 8-mm. round hole to rib | 31  | 1 rib   |
| 6-mm. round hole to rib | 41  | 10 ribs |
| 9-mm. round hole to rib | 43  | 2 ribs  |
| 5-mm. round hole to rib | end | 3 ribs  |

Close examination of the jigs brings out the following points:

1. Stratification of the material into layers of varying richness takes

place almost immediately after it reaches the tray. This is helped by the strong pulsion effect on the first hutch, where suction is minimized by the fineness of the bed of free galena, which lies in a heavy mass on the screen.

2. In order to remove all the material containing lead, both as free galena and middling, a bed is required that gradually decreases in specific gravity, with increasing size of grains. This was found to be most effectively obtained by arranging the tray with an occasional row of large holes, to take out the bedding in steps.

3. The bed may be likened to a screen of varying size holes, this size depending on the size of grains. Any section should be just long enough to give a good "screening efficiency" on such material as will pass through it. For instance, a bed composed of grains of specific gravity 4.0 will hutch a product assaying about 40 per cent. lead. If it be found that all the 40 per cent. material present will pass through in a length of 2 ft., there is a loss of efficiency by extending this particular bed any farther

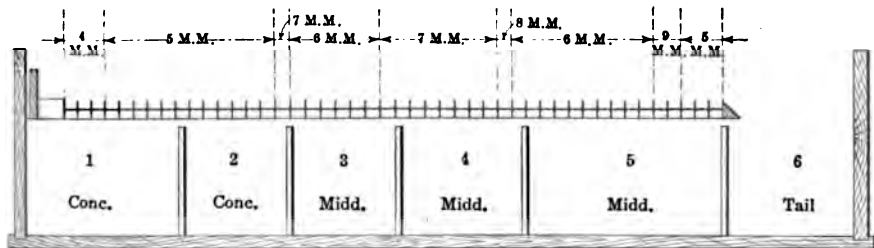


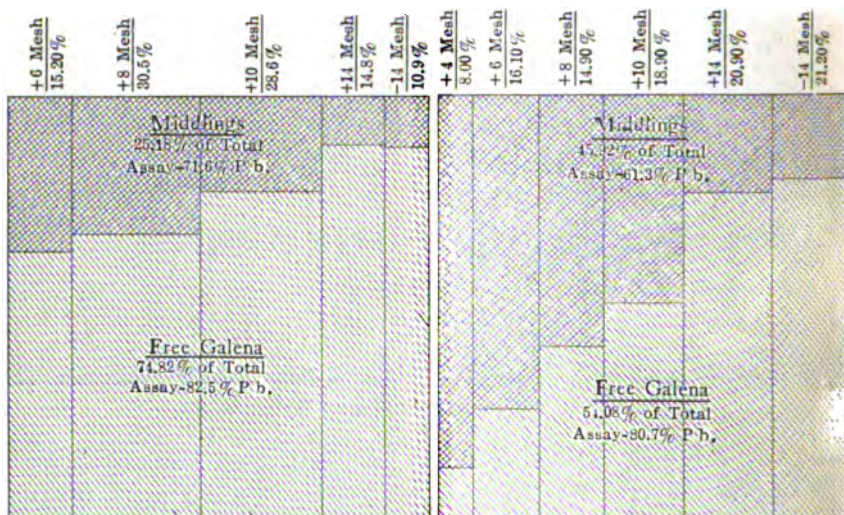
FIG. 4.—SYSTEM OF SCREENS USED ON HANCOCK JIG.

than 2 ft. A row of large holes at this point will allow the heavier particles to pass through, leaving a bed of lighter specific gravity for the next section of tray. A bed which changes rapidly in character will therefore allow all parts of the jig to do equal work, and give maximum efficiency.

4. There are two distinct operations to be considered. First, the making of clean concentrates, and second, the making of clean tailing. These are justly considered apart, and of the two the latter is by far the more difficult. The specific gravity of galena is 7.40 and of gangue 2.80, and while the separation of these two minerals is easy, the material received on the jig is made up of grains ranging from one extreme to the other. By sorting the products of the jig (Table 4) it is found that concentrates are made up of grains of free galena of grade 82.3 per cent. lead, and middlings of 60.4 per cent. lead. Specific gravities are 7.00 and 5.20 respectively; a difference of 1.80 tailings are made up of middlings of grade 4.91 per cent. or specific gravity 2.95, and gangue of grade 0.14 per cent., or specific gravity 2.82, a difference of 0.13. These figures

illustrate the wider range of specific gravity permissible in concentrates, and therefore the relative ease of separation of the high-grade material.

5. The small amount of free galena present, as shown in Table 2, also helps to simplify its separation, and it would appear to be good practice to hutch the free galena in as short a distance as possible, reserving the greater length of the tray for the more difficult work of separating middling from tailing. An attempt to do this was very successful, and concentrates are now made on only two hutches, occupying 80 in. of screen length. The larger part is caught in the first hutch, the second acting as a safeguard to prevent free galena being carried over into the middling hutches. As a general rule, the tonnage of product from the second hutch is very small.



6. The separation on the final section of the tray, or the part over the fifth hutch, must necessarily be rough, owing to the low grade of the middling remaining, and the small difference in specific gravity between it and the free gangue. The bed consists of large-size grains of low-grade middling, and the heaviest tonnage of hutched product is made here. At the end of the fifth hutch, two ribs of holes, large enough to take out all the remaining bedding, are used.

7. Variations in grade of the feed introduce complications that must be allowed for in the screen layout. An excess of bedding material, due to higher-grade ore than usual, causes an accumulation of heavy grains that cannot pass through the screens, while low-grade ore will allow the bed to become too light, and the hutched product will contain too much gangue.

By arranging the screen to take out the bedding in steps, as mentioned before, such an accumulation cannot extend very far, but extreme varia-

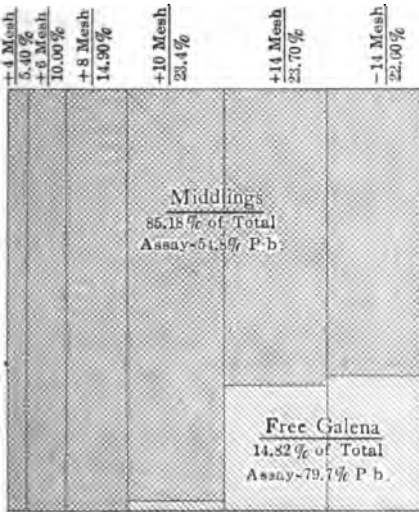


FIG. 7.—SCREEN ANALYSIS OF THIRD HUTCH PRODUCT.

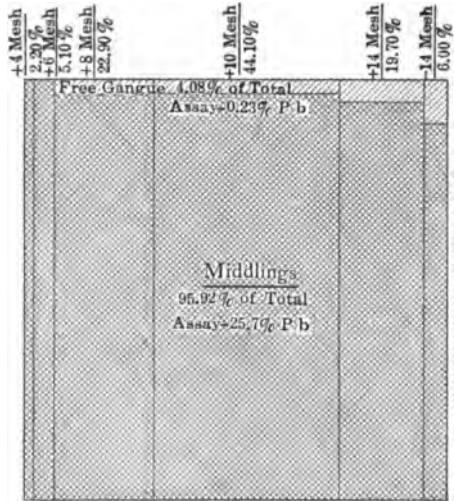


FIG. 8.—SCREEN ANALYSIS OF FOURTH HUTCH PRODUCT.

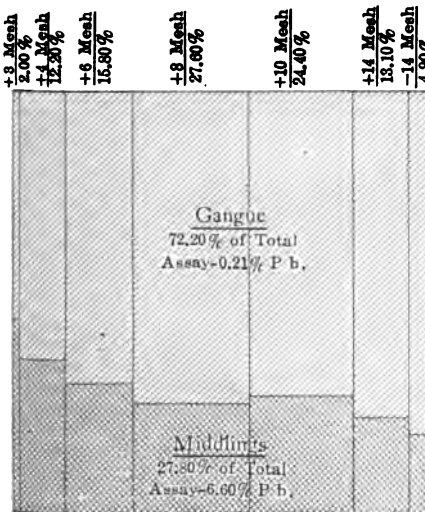


FIG. 9.—SCREEN ANALYSIS OF FIFTH HUTCH PRODUCT.

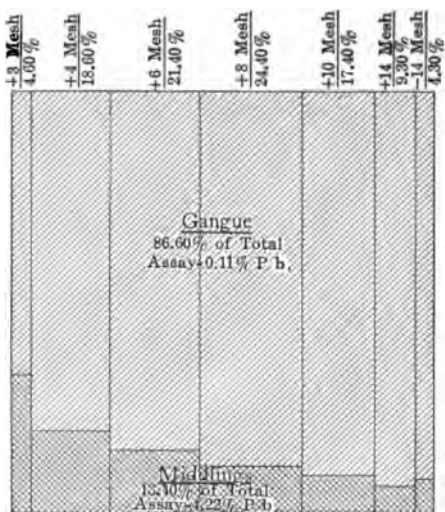


FIG. 10.—SCREEN ANALYSIS OF JIG TAILING.

tions are best taken care of by the operator, who shovels off excess bed, and plugs the large holes in the screen when bedding material is scarce.

8. Wire cloth and punched plate were both tried for the jig screens,

| Mesh                            | Per Cent. Weight | Assay, Per Cent. Lead | Per Cent. of Total Lead Content | Per Cent. of Weight on Each Mesh |       |       | Per Cent. of Total Lead Content |       |       |
|---------------------------------|------------------|-----------------------|---------------------------------|----------------------------------|-------|-------|---------------------------------|-------|-------|
|                                 |                  |                       |                                 | Free Galena                      | Midd. |       | Free Galena                     | Midd. |       |
| On 3.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 4.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 6.....                       | 15.2             | 80.3                  | 15.29                           | 63.0                             | 37.0  | 9.58  | 82.1                            | 76.9  | 64.50 |
| On 8.....                       | 30.5             | 78.6                  | 30.08                           | 67.2                             | 32.8  | 20.50 | 82.7                            | 70.4  | 70.65 |
| On 10.....                      | 28.6             | 79.7                  | 28.60                           | 77.2                             | 22.8  | 22.10 | 81.7                            | 70.3  | 79.62 |
| On 14.....                      | 14.8             | 81.2                  | 15.09                           | 88.3                             | 11.7  | 13.07 | 83.1                            | 66.7  | 90.42 |
| Through 14.....                 | 10.9             | 79.8                  | 10.94                           | 87.8                             | 12.2  | 9.57  | 82.0                            | 65.1  | 90.00 |
| Total.....                      | 100.0            | 79.7                  | 100.00                          | 74.8                             | 25.2  | 74.82 | 82.5                            | 71.6  | 77.40 |
| 7. 2d Hutch                     |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 3.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 4.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 6.....                       | 8.0              | 68.2                  | 7.60                            | 12.0                             | 88.0  | 0.96  | 79.0                            | 66.8  | 13.90 |
| On 8.....                       | 16.1             | 68.9                  | 15.45                           | 26.0                             | 74.0  | 4.18  | 80.2                            | 65.1  | 30.15 |
| On 10.....                      | 14.9             | 71.3                  | 14.80                           | 41.0                             | 59.0  | 6.11  | 80.6                            | 65.0  | 46.30 |
| On 14.....                      | 18.9             | 69.5                  | 18.30                           | 51.3                             | 48.7  | 9.70  | 81.7                            | 56.8  | 60.20 |
| Through 14.....                 | 20.9             | 74.7                  | 21.75                           | 77.2                             | 22.8  | 16.15 | 80.9                            | 53.4  | 83.75 |
| Total.....                      | 100.0            | 71.8                  | 100.00                          | 80.1                             | 19.9  | 16.98 | 80.0                            | 53.2  | 85.85 |
| 8. 3d Hutch                     |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 3.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 4.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 6.....                       | 5.4              | 59.9                  | 5.54                            | 100.0                            |       |       |                                 |       |       |
| On 8.....                       | 10.0             | 56.4                  | 9.98                            | 100.0                            |       |       |                                 |       |       |
| On 10.....                      | 14.9             | 59.8                  | 15.25                           | 100.0                            |       |       |                                 |       |       |
| On 14.....                      | 23.4             | 59.2                  | 23.68                           | 2.3                              | 97.7  | 0.54  | 80.1                            | 58.7  | 3.12  |
| Through 14.....                 | 23.7             | 63.3                  | 25.65                           | 29.8                             | 70.2  | 7.06  | 70.3                            | 56.5  | 37.80 |
| Total.....                      | 100.0            | 58.5                  | 100.00                          | 32.0                             | 68.0  | 7.22  | 70.8                            | 58.2  | 49.60 |
| Per Cent. of Total Lead Content |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 3.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 4.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 6.....                       | 5.4              | 59.9                  | 5.54                            | 100.0                            |       |       |                                 |       |       |
| On 8.....                       | 10.0             | 56.4                  | 9.98                            | 100.0                            |       |       |                                 |       |       |
| On 10.....                      | 14.9             | 59.8                  | 15.25                           | 100.0                            |       |       |                                 |       |       |
| On 14.....                      | 23.4             | 59.2                  | 23.68                           | 2.3                              | 97.7  | 0.54  | 80.1                            | 58.7  | 3.12  |
| Through 14.....                 | 23.7             | 63.3                  | 25.65                           | 29.8                             | 70.2  | 7.06  | 70.3                            | 56.5  | 37.80 |
| Total.....                      | 100.0            | 58.5                  | 100.00                          | 32.0                             | 68.0  | 7.22  | 70.8                            | 58.2  | 49.60 |
| Per Cent. of Total Lead Content |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 3.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 4.....                       |                  |                       |                                 |                                  |       |       |                                 |       |       |
| On 6.....                       | 5.4              | 59.9                  | 5.54                            | 100.0                            |       |       |                                 |       |       |
| On 8.....                       | 10.0             | 56.4                  | 9.98                            | 100.0                            |       |       |                                 |       |       |
| On 10.....                      | 14.9             | 59.8                  | 15.25                           | 100.0                            |       |       |                                 |       |       |
| On 14.....                      | 23.4             | 59.2                  | 23.68                           | 2.3                              | 97.7  | 0.54  | 80.1                            | 58.7  | 3.12  |
| Through 14.....                 | 23.7             | 63.3                  | 25.65                           | 29.8                             | 70.2  | 7.06  | 70.3                            | 56.5  | 37.80 |
| Total.....                      | 100.0            | 58.5                  | 100.00                          | 32.0                             | 68.0  | 7.22  | 70.8                            | 58.2  | 49.60 |

9. 4th Hutch

| Mesh            | Per Cent. Weight | Assay, Per Cent. Lead | Per Cent. of Total Lead Content | Per Cent. of Weight on Each Mesh |             | Per Cent. of Total Weight |             | Assay, Per Cent. Lead |             | Per Cent. of Lead on Each Mesh |             | Per Cent. of Total Lead Content |             |
|-----------------|------------------|-----------------------|---------------------------------|----------------------------------|-------------|---------------------------|-------------|-----------------------|-------------|--------------------------------|-------------|---------------------------------|-------------|
|                 |                  |                       |                                 | Midd.                            | Free Gangue | Midd.                     | Free Gangue | Midd.                 | Free Gangue | Midd.                          | Free Gangue | Midd.                           | Free Gangue |
| On 3.....       | 2.2              | 37.0                  | 3.30                            | 100.0                            | .....       | 2.20                      | .....       | 37.0                  | .....       | 100.00                         | .....       | 3.30                            | .....       |
| On 4.....       | 5.1              | 29.1                  | 6.01                            | 100.0                            | .....       | 5.10                      | .....       | 29.1                  | .....       | 100.00                         | .....       | 6.01                            | .....       |
| On 6.....       | 22.9             | 27.1                  | 25.14                           | 96.2                             | 3.8         | 22.03                     | 0.87        | 28.2                  | 0.18        | 99.975                         | 0.025       | 25.14                           | 0.006       |
| On 8.....       | 44.1             | 22.6                  | 40.22                           | 96.6                             | 3.4         | 42.59                     | 1.51        | 23.4                  | 0.20        | 99.966                         | 0.034       | 40.21                           | 0.014       |
| On 10.....      | 19.7             | 25.2                  | 20.13                           | 94.7                             | 5.3         | 18.66                     | 1.04        | 26.6                  | 0.24        | 99.950                         | 0.050       | 20.12                           | 0.010       |
| On 14.....      | 6.0              | 21.4                  | 5.20                            | 89.0                             | 11.0        | 5.34                      | 0.66        | 24.0                  | 0.36        | 99.815                         | 0.185       | 5.19                            | 0.009       |
| Through 14..... |                  |                       |                                 |                                  |             |                           |             |                       |             |                                |             |                                 |             |
| Total.....      | 100.0            | 24.7                  | 100.00                          | 95.9                             | 4.1         | 95.92                     | 4.08        | 25.7                  | 0.23        | 99.961                         | 0.039       | 99.96                           | 0.039       |

10. 5th Hutch

| Mesh            | Per Cent. Weight | Assay, Per Cent. Lead | Per Cent. of Total Lead Content | Per Cent. of Weight on Each Mesh |             | Per Cent. of Total Weight |             | Assay, Per Cent. Lead |             | Per Cent. of Lead on Each Mesh |             | Per Cent. of Total Lead Content |             |
|-----------------|------------------|-----------------------|---------------------------------|----------------------------------|-------------|---------------------------|-------------|-----------------------|-------------|--------------------------------|-------------|---------------------------------|-------------|
|                 |                  |                       |                                 | Midd.                            | Free Gangue | Midd.                     | Free Gangue | Midd.                 | Free Gangue | Midd.                          | Free Gangue | Midd.                           | Free Gangue |
| On 3.....       | 2.0              | 1.15                  | 1.16                            | 46.0                             | 54.0        | 0.92                      | 1.06        | 2.32                  | 0.15        | 93.05                          | 6.95        | 1.08                            | 0.08        |
| On 4.....       | 12.2             | 1.63                  | 9.95                            | 36.1                             | 63.9        | 4.40                      | 7.80        | 4.07                  | 0.24        | 90.49                          | 9.51        | 9.01                            | 0.94        |
| On 6.....       | 15.8             | 1.56                  | 12.38                           | 30.5                             | 69.5        | 4.80                      | 11.00       | 4.66                  | 0.20        | 91.07                          | 8.93        | 11.27                           | 1.11        |
| On 8.....       | 27.6             | 2.20                  | 30.47                           | 26.0                             | 74.0        | 7.15                      | 20.45       | 7.96                  | 0.18        | 93.94                          | 6.06        | 28.62                           | 1.86        |
| On 10.....      | 24.4             | 2.36                  | 28.90                           | 27.4                             | 72.6        | 6.67                      | 17.73       | 7.97                  | 0.25        | 92.37                          | 7.63        | 26.68                           | 2.22        |
| On 14.....      | 13.1             | 2.04                  | 13.42                           | 22.7                             | 77.3        | 2.96                      | 10.14       | 8.25                  | 0.22        | 91.66                          | 8.34        | 12.30                           | 1.12        |
| Through 14..... | 4.9              | 1.67                  | 3.72                            | 18.4                             | 81.6        | 0.90                      | 4.00        | 7.42                  | 0.18        | 90.28                          | 9.72        | 3.35                            | 0.37        |
| Total.....      | 100.0            | 2.00                  | 100.00                          | 27.8                             | 72.2        | 27.80                     | 72.20       | 6.60                  | 0.21        | 92.31                          | 7.69        | 92.31                           | 7.69        |

11. Tailings

| Mesh            | Per Cent. Weight | Assay, Per Cent. Lead | Per Cent. of Total Lead Content | Per Cent. of Weight on Each Mesh |             | Per Cent. of Total Weight |             | Assay, Per Cent. Lead |             | Per Cent. of Lead on Each Mesh |             | Per Cent. of Total Lead Content |             |
|-----------------|------------------|-----------------------|---------------------------------|----------------------------------|-------------|---------------------------|-------------|-----------------------|-------------|--------------------------------|-------------|---------------------------------|-------------|
|                 |                  |                       |                                 | Midd.                            | Free Gangue | Midd.                     | Free Gangue | Midd.                 | Free Gangue | Midd.                          | Free Gangue | Midd.                           | Free Gangue |
| On 3.....       | 4.6              | 1.04                  | 7.26                            | 32.8                             | 67.2        | 1.510                     | 3.090       | 2.92                  | 0.13        | 91.67                          | 8.33        | 6.655                           | 0.605       |
| On 4.....       | 18.6             | 0.67                  | 18.90                           | 19.3                             | 80.7        | 3.585                     | 15.015      | 3.07                  | 0.10        | 88.00                          | 12.00       | 16.630                          | 2.270       |
| On 6.....       | 21.4             | 0.80                  | 20.70                           | 14.8                             | 85.2        | 3.165                     | 18.235      | 3.84                  | 0.08        | 89.20                          | 10.80       | 18.490                          | 2.210       |
| On 8.....       | 24.4             | 0.63                  | 23.23                           | 10.9                             | 89.1        | 2.660                     | 21.740      | 5.12                  | 0.08        | 88.67                          | 11.33       | 20.625                          | 2.605       |
| On 10.....      | 17.4             | 0.63                  | 16.74                           | 8.8                              | 91.2        | 1.530                     | 15.870      | 6.60                  | 0.06        | 91.40                          | 8.60        | 15.300                          | 1.440       |
| On 14.....      | 9.3              | 0.69                  | 9.70                            | 6.6                              | 93.4        | 0.615                     | 8.685       | 6.24                  | 0.30        | 59.30                          | 40.70       | 5.750                           | 3.950       |
| Through 14..... | 4.3              | 0.53                  | 3.47                            | 7.8                              | 92.2        | 0.335                     | 3.965       | 4.80                  | 0.20        | 65.50                          | 34.50       | 2.280                           | 1.190       |
| Total.....      | 100.0            | 0.66                  | 100.00                          | 13.4                             | 86.6        | 13.400                    | 86.600      | 4.22                  | 0.11        | 85.73                          | 14.27       | 85.730                          | 14.270      |

with a final decision in favor of the latter. While it is sometimes claimed that wire cloth gives less trouble by blinding, it was found in these tests that there was very little difference in this respect, while punched plate had the advantage of being more easily cleaned.

The nature of the products obtained with the screen system given in Table 5, are shown in detail in Tables 6 to 11 and Figs. 5 to 10. The first two hutches produce good concentrate, the third and fourth a rich middling, and the fifth a low-grade middling containing a large proportion of free gangue. Analysis of the tailing shows some middling escaping. The lead in this occurs as very small pieces sticking to the large pieces of gangue, or included in them, and owing to its low specific gravity it is doubtful whether this low-grade middling could be saved without recrushing the whole tailing. Table 12 shows results of a tonnage test on the jig, equipped with the screen system given in Table 5.

TABLE 12.—*Work of Hancock Jig Under Present Conditions*

| Material          | Tons per 24 Hr. | Assay, Per Cent.<br>Lead | Tons Lead per 24<br>Hr. |
|-------------------|-----------------|--------------------------|-------------------------|
| Feed.....         | 505.0           | 4.74                     | 23.95                   |
| Concentrates..... | 7.7             | 75.00                    | 5.79                    |
| Middlings.....    | 195.0           | 8.16                     | 15.91                   |
| Tailing.....      | 302.3           | 0.74                     | 2.25                    |

The rolls have a feed of 195 tons per 24 hr., of which 58 per cent., or 113 tons, is crushed fine enough to pass through 2-mm. screens. The remaining 82 tons returns to the jig with 423 tons of original ore. The lighter load on the jig enables it to do better separation, and a cleaner concentrate and lower tailing are possible.

Subsequently the tonnage of ore treated in the mill was raised from 1,800 to 2,200 tons per day without extra jig equipment, and with lower tailings than previously, the gain being almost entirely due to the more efficient work done by the jigs.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Experiments in the Recovery of Tungsten and Gold in the Murray District, Idaho\*

BY ROBERT RHEA GOODRICH,† AND NORMAN E. HOLDEN,‡ MOSCOW, IDAHO

(St. Louis Meeting, October, 1917)

THERE is a small area about Murray where bedded gold quartz veins occur in Prichard slate. The ore taken from the upper levels, during the early days, was free milling and was treated by stamp milling and amalgamation. There was likewise a considerable placer working. The report is current that tungsten (scheelite) was encountered, which interfered with the amalgamation in the stamp mills and which, in placering, collected in the riffles in the sluice boxes and clogged them. Consequently, in the quartz mines, the scheelite, when encountered, was rejected, being thrown into the waste, sometimes inside the mine and at other times outside on the dump. The free surface ores became worked out and as the ore in depth became pyritic and rebellious to amalgamation the mines closed down.

Last spring tungsten ore brought a high price, near \$80 per unit, 60 per cent.  $WO_3$  and up. This so stimulated the output of tungsten ore that old dumps were explored and waste stored in the mines was gone over. Tungsten was likewise mined from virgin ground. So the district produced quite heavily for a time.

The condition now is different. The price is not so high, about \$20 per unit  $WO_3$ , which makes profits questionable when mining in new ground. The accumulations from early day mining have been exhausted and the output has greatly declined.

The Golden Chest mine was the heaviest producer of tungsten. The months of June and July were spent at this mine studying the problem. Since the mine had produced both gold and tungsten, we looked forward to the problem of recovering both from what might be called a gold-tungsten ore. A study of the Golden Chest vein showed this not to be so there. While gold ore and tungsten ore come from the same vein, the tungsten ore is confined to a more or less disturbed or distorted portion

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\*This research was partly done at the Golden Chest Mine, Murray, Idaho, during the summer months of 1916, and completed in the laboratory of the University of Idaho, Moscow, Idaho, during the winter of 1916-17.

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some middlings which were reground and returned to the system, and concentrates. There was produced in the hutch a product which was the feed to the 60-mesh hand-jig. The products on the sieve of the 60-mesh hand-jig were also clean tailings, some middlings which were reground and returned to the system, and concentrates. There was produced in the hutch a product containing about 18 per cent.  $\text{WO}_3$ . This hutch product of the 60-mesh hand-jig was not treated but was allowed to accumulate. The experiment was made in order to determine the best method of recovering the scheelite from this product. The losses in the tailings rejected from the 8-mesh hand-jig and the 60-mesh hand-jig were made practically nil, consequently the entire milling loss occurred in the treat-

TABLE 1—PART 2

|     |                    | 10 grams of Concentrates (Raw)<br>roasted and separated by magnet into Purified $\text{CaWO}_4$ Concentrate and Magnetite Iron Sulphide |              |              |              |             |                  |                  |                     |                |                |                         |                  |              |                  |              |                  |              |                  |              |                  |
|-----|--------------------|---|--------------|--------------|--------------|-------------|------------------|------------------|---------------------|----------------|----------------|-------------------------|------------------|--------------|------------------|--------------|------------------|--------------|------------------|--------------|------------------|
|     |                    | Purified $\text{CaWO}_4$ Concentrate  |              |              |              |             |                  |                  |                     |                |                | Magnetite Iron Sulphide |                  |              |                  |              |                  |              |                  |              |                  |
| No. | Size               | Chemical Analysis   |              |              |              |             |                  |                  |                     |                |                | Physical Analysis       |                  |              |                  |              |                  |              |                  |              |                  |
|     |                    | Fe - percent  | Si - percent | Al - percent | Ca - percent | W - percent | As - weight, mg. | Mo - weight, mg. | Other - weight, mg. | Loss - percent | Loss - percent | Weight, gms.            | Specific gravity | Weight, gms. | Specific gravity | Weight, gms. | Specific gravity | Weight, gms. | Specific gravity | Weight, gms. | Specific gravity |
| 1   | 1/2"               | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 2   | 3/8"               | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 3   | 3/16"              | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 4   | 1/8"               | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 5   | 1/16"              | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 6   | 1/32"              | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 7   | 1/64"              | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 8   | 1/128"             | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 9   | 1/256"             | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 10  | 1/512"             | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 11  | 1/1024"            | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 12  | 1/2048"            | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 13  | 1/4096"            | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 14  | 1/8192"            | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 15  | 1/16384"           | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 16  | 1/32768"           | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 17  | 1/65536"           | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 18  | 1/131072"          | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 19  | 1/262144"          | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 20  | 1/524288"          | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 21  | 1/1048576"         | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 22  | 1/2097152"         | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 23  | 1/4194304"         | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 24  | 1/8388608"         | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 25  | 1/16777216"        | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 26  | 1/33554432"        | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 27  | 1/67108864"        | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 28  | 1/134217728"       | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 29  | 1/268435456"       | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 30  | 1/536870912"       | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 31  | 1/1073741824"      | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 32  | 1/2147483648"      | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 33  | 1/4294967296"      | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 34  | 1/8589934592"      | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 35  | 1/17179869184"     | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 36  | 1/34359738368"     | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 37  | 1/68719476736"     | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 38  | 1/137438953472"    | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 39  | 1/274877906944"    | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 40  | 1/549755813888"    | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 41  | 1/1099511627776"   | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 42  | 1/2199023255552"   | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 43  | 1/4398046511104"   | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 44  | 1/8796093022208"   | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 45  | 1/17592186044416"  | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 46  | 1/35184372088832"  | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 47  | 1/70368744177664"  | 75.0  | 0.0          | 0.0          | 0.0          | 5.0         | 0.0              | 0.0              | 0.0                 | 0.0            | 0.0            | 10.0                    | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              | 10.0         | 5.0              |
| 48  | 1/140737488355328" | 75.   |              |              |              |             |                  |                  |                     |                |                |                         |                  |              |                  |              |                  |              |                  |              |                  |

lite is that scheelite does not tend to float. The tungsten recovery in the concentrates (raw) from the total Wilfley feed was 91.6 per cent.

Certain buyers of tungsten concentrates asked for a  $\text{WO}_3$  content greater than 60 per cent. and for a sulphur content less than 1 per cent. The 8-mesh hand-jig concentrates invariably fulfilled the condition as to  $\text{WO}_3$ , but not the condition as to sulphur. The 60-mesh hand-jig concentrates invariably had low  $\text{WO}_3$  and high sulphur contents. All the Wilfley concentrates had low  $\text{WO}_3$  and high sulphur contents. Two methods presented themselves for purifying the concentrates by removing the pyrite:



FIG. 1.—VEIN NOT IN TUNGSTEN ZONE, GOLD BEARING.

- (a) Flotation.
- (b) Roasting for magnetism followed by magnetic separation.

#### *Flotation<sup>1</sup>*

The underflow (purified tungsten product) gave  $\text{WO}_3$ , 70.9 per cent.; iron, 2.5 per cent.; sulphur, 2.7 per cent. The overflow gave  $\text{WO}_3$ , 11.8 per cent.; iron, 36.7 per cent.; sulphur 41.0 per cent. The purified tungsten product was not bad, yet a much cleaner product was desired. The overflow carried considerable tungsten, which was a loss and, moreover, the pyritic tungsten concentrates, before they could be fed to the flotation

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<sup>1</sup>Only one test was made. By further experimenting better results might be secured.

machine, were required to be reground. The objections to this method are:

1. The purified tungsten concentrates are not sufficiently clean.
2. There is considerable loss of tungsten in the overflow.
3. Fine grinding is required.

*Roasting for Magnetism followed by Magnetic Separation*

Ten grams of pyritic tungsten concentrates were roasted in an assayer's gasoline-fired muffle furnace in a 2½-in. roasting dish. The



FIG. 2.—VEIN IN THE TUNGSTEN ZONE. POCKET OF TUNGSTEN (SCHEELITE) MARKED S.

temperature was measured by a thermo-electric pyrometer. Preliminary roasts to determine the best roasting temperature were made at 700°, 800°, 850°, and 900° C. The temperature of 850° C. gave the best results and this temperature was used in making the final complete test. The roasting dish was not covered and the roast was stirred occasionally. Flaming ceased slightly before the completion of the roast. It was believed that this method of roasting would produce conditions similar to those that would exist in practice when treating the material in quantity.

In the final test, two series of roasts were made. The temperature of 850° C. was used in both series. The first series was made uncovered and the material was stirred. The magnetic separation of the roasted mate-

rial was made, producing purified  $\text{CaWO}_4$  concentrates and magnetic iron sulphide. The purified  $\text{CaWO}_4$  concentrates were weighed, then the entire amount was ground fine and used in making the determinations of  $\text{WO}_3$ , iron, and sulphur. Gold values were desired in the purified  $\text{CaWO}_4$  concentrates and in the magnetic iron sulphide. These values had not been determined and the materials were used up. Consequently, it was necessary to make a second series of roasts in order to provide materials for use in the determination of the gold. Since it had been noted that in making the roasts there was some loss of material by decrepitation, especially with the coarse sizes, it was decided to make this second series of roasts closely covered and not to stir the material at all.

The second series of roasts was made closely covered and the decrepitation loss was thus entirely eliminated, facilitating the accounting for material. The roasts were not uncovered until they were out of the furnace and cold. The results of roasting covered were very satisfactory. Magnetic separation of the roasted material was made in the same way as in the first series of roasts, producing purified  $\text{CaWO}_4$  concentrates and magnetic iron sulphide. The purified  $\text{CaWO}_4$  concentrates produced, when roasting had been carried on uncovered (first series of roasts), had a slight reddish tinge due to the formation of some  $\text{Fe}_2\text{O}_3$  that remained with and discolored the  $\text{CaWO}_4$  which would otherwise have been white. When the roast that had been made covered (second series) was subjected to magnetic separation, the magnetic iron sulphide appeared more strongly magnetic than that produced when roasting uncovered. The result was that the magnetic separation of this second series of roasts was accomplished with greater ease than that of the first series. The resultant purified  $\text{CaWO}_4$  concentrates were white and had no reddish discoloration as did the product from the first series of roasts. The two products of this second series of roasts, the purified  $\text{CaWO}_4$  concentrates and the magnetic iron sulphide, were then used for determining the gold content of each.

On examination under the microscope, the purified  $\text{CaWO}_4$  concentrates resulting from the second series of roasts appeared as clean as the product of the first series. If there was any difference, the product of the second series appeared slightly cleaner. The weight of the purified  $\text{CaWO}_4$  concentrates, resulting from the covered roast, was used as a basis of balancing quantities. The gold was likewise determined in the products of the second series of roasts, while the  $\text{WO}_3$ , iron, and sulphur were determined in the products of the first series. Decrepitation loss was thus eliminated. It is believed that if there is any error introduced by thus combining the data obtained from the two series of roasts, it is very small and on the safe side, for if determinations of the  $\text{WO}_3$ , iron, and sulphur had been made on the products of the second series of roasts, the  $\text{WO}_3$  would probably have been slightly higher and the iron and the sul-

phur slightly lower than the values reported. It is to be noted that on the finest size, through-200-mesh, where the decrepitation loss was known to be practically nil when roasting uncovered, there is practically no difference in the weights of the purified  $\text{CaWO}_4$  concentrates from the two series of roasts.

In making the magnetic separation, two magnets were used:

- (a) Roughing magnet.<sup>2</sup>
- (b) Finishing magnet.

The roughing magnet removed easily the bulk of the magnetic material. After using the roughing magnet, there remained a small amount of weakly magnetic material which required for removal a stronger magnet.

The purified  $\text{CaWO}_4$  concentrates have a  $\text{WO}_3$  content of 64 to 78 per cent., which is not far below that of the pure mineral, 80.6 per cent. The recovery of  $\text{WO}_3$  in the purified  $\text{CaWO}_4$  concentrates, in all but the through-200-mesh product, is greater than 95 per cent. of the  $\text{WO}_3$  in the concentrates (raw). In purifying the through-200-mesh size, there was a tendency for more  $\text{CaWO}_4$  to be entangled in the magnetic material and to be carried with it, and in treating that material, there was recovery in the purified  $\text{CaWO}_4$  concentrates only 93.1 per cent. of the  $\text{WO}_3$  in the concentrates (raw). There is thus but a small loss of  $\text{WO}_3$  in the magnetic iron sulphide. The iron and sulphur are both low in the purified  $\text{CaWO}_4$  concentrates. The iron in all but the two finest sizes is around 1 per cent., and the sulphur about 0.1 per cent. in all sizes.

The net recovery of  $\text{WO}_3$  in the double operation, gravity concentration and magnetic separation, is the product of the two separate recoveries. This is recorded in the column of the tabulation headed, " $\text{WO}_3$  recovered in purified  $\text{CaWO}_4$  concentrates, per cent. of  $\text{WO}_3$  in feed." On all but the two finest sizes, this net recovery is practically 90 per cent. or greater, and on the two finest sizes 76 to 83 per cent.

### *The Saving of Gold*

As was stated at the start, we are not treating a gold ore in this experiment. The material treated is a tungsten material containing a small amount of gold. The distribution of the gold in the different products of this experiment has been investigated in order to make the experiment complete. The magnetic iron sulphide separated, although carrying an ounce of gold per ton, is of small importance commercially because of its small tonnage. For instance, suppose 4 tons of purified  $\text{CaWO}_4$  concentrates were produced in a month. There would be produced less than 2 tons of magnetic iron sulphide carrying about 1 oz. of gold per ton and of a gross value of about \$40. Consequently the investigation of the

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<sup>2</sup>Separating magnet, manufactured and sold by A. H. Kidney, 255 Park Avenue, Orange, N. J.

distribution of the gold in the products is interesting only from a technical standpoint. The profits must be derived from the output of purified  $\text{CaWO}_4$  concentrates.

### *Epitome*

1. In the Golden Chest vein, the tungsten mineral, scheelite, occurs in rich segregations, in a more or less disturbed or distorted portion of the vein known as the tungsten zone, barren of gold and pyrite-free.

2. Since the tungsten mineral in the vein occurs in these rich segregations, a considerable portion of the ore is sacked in the stope and sent out to a cleaning floor, where it is brought up to a shipping grade by hand-picking.

3. In breaking down the ore in the mine, some fine tungsten mineral becomes mixed with waste rock and some pyritic material carrying small gold values. The tungsten mineral is recovered from this material (mill feed) by concentration.

4. Gravity concentration produces pyritic tungsten concentrates, 53.7 per cent.  $\text{WO}_3$ , with high pyrite content. This is not a shipping grade.

5. The objections to flotation, as a means of purifying the pyritic tungsten concentrates, are: The purified tungsten concentrates are not sufficiently clean; there is considerable loss of tungsten in the overflow; and fine grinding is required.

6. The pyritic tungsten concentrates, when treated by roasting for magnetism followed by magnetic separation, yield purified  $\text{CaWO}_4$  concentrates of shipping grade,  $\text{WO}_3$  greater than 60 per cent., and sulphur less than 1 per cent. On all but the two finest sizes, the net recovery, expressed as per cent. of  $\text{WO}_3$  in the mill feed, is practically 90 per cent. or greater, and on the two finest sizes, 76 to 83 per cent.

7. The investigation of the distribution of gold in the products is interesting only from a technical standpoint and not from a commercial. The profits must be derived from the output of purified  $\text{CaWO}_4$  concentrates.



**DISCUSSION OF THIS PAPER IS INVITED.** It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The New Jersey Zinc Co.'s Franklin Laboratory

BY D. JENKINS, FRANKLIN FURNACE, N. J.

(St. Louis Meeting, October, 1917)

THE Franklin Laboratory was designed mainly for the analysis of the products from the two concentrating mills situated at Franklin and Sterling Hill, the most important determinations being the zinc, iron, manganese, lime, moisture and silica contents of the ores. Within the last 4 years, however, it has been found advisable to examine most of our supplies, and therefore additional space and equipment was allotted for the examination of such substances as oils, greases, soaps, alloys, fuels, paints, explosives and water.

The building as shown in elevation (Fig. 1) and plan (Fig. 2) consists of a main or routine laboratory, an experimental laboratory, a water

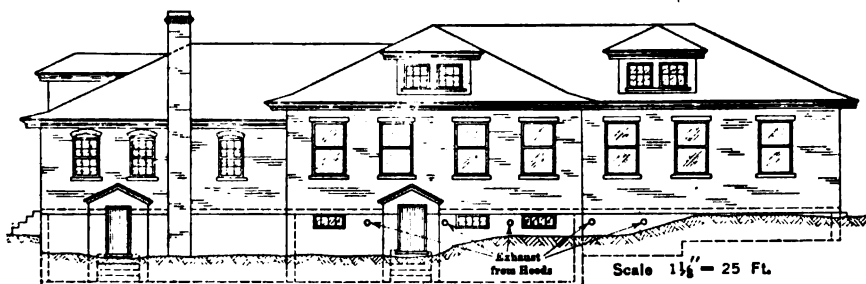


FIG. 1.—WEST ELEVATION.

analysis room and the other accessories such as stock room, library, sampling and drying rooms and office. The main laboratory is further divided into a balance room and zinc analysis room.

As the greater part of our determinations are volumetric, good light is a very important factor and with this in view the interior walls of the building were constructed of white enameled brick laid in Keene cement. These also have the additional advantage that they give a clean and attractive appearance to the room. The windows are large, being 76 by 44 in. and even on very dark days the light is good throughout the whole laboratory. Our zinc titration requires a constant light and as, on many days, especially in winter, the light is variable, we installed two Artificial

Daylight lamps and have found that they meet all our requirements. They also make it possible to carry out titrations at night if necessary.

On referring again to Fig. 2, one can obtain an idea of the arrangement of the desks, sinks, cases and hoods. All of these have been designed with the idea of expediting the work as much as possible with the least confusion. The hoods shown at the center and running the whole length of the room were designed by R. M. Catlin, and as they are somewhat novel a few words in explanation of their construction may be apropos. They consist of a series of small compartments, there being in this case 50. Each compartment is about 27 in. long, 8 in. wide and 8 in. high and is

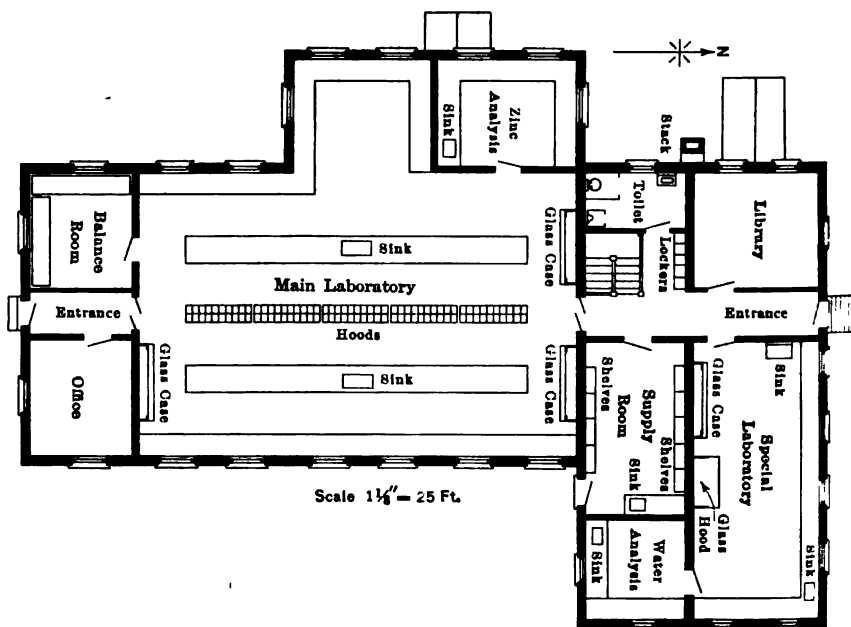


FIG. 2.—FLOOR PLAN.

made of Alberene stone. Each hood accommodates two sand baths, and all the gases arising from evaporation are taken off at the center of the hood and conducted outside of the building by means of a slight vacuum produced by an air aspirator placed in the mouth of a 6-in. Orangeburg fiber pipe, which leads from the hoods. One aspirator exhausts the gases from 10 compartments. Each compartment is supplied with two sliding doors which fit into grooves cut in the sides of the hood. By means of a notch cut in these grooves, the doors may be partly opened if necessary. Heat is furnished by means of a series of Bunsen burners placed under each sand bath. A better idea of the construction of the hoods may be obtained from the attached photograph and drawings.

In our old laboratory we had experienced great difficulty in keeping our wooden floor in good condition, so in our new building we treated the wood with aniline black and up to the present time the floors have remained in excellent condition despite the fact that in many places they have been subjected unintentionally to the action of concentrated acids.

Adjoining the main laboratory is a room set apart for zinc titration only. This arrangement has many advantages, the chief being that the atmosphere of the main room is not contaminated with ammonia fumes, as all of our zinc titrations are made in ammoniacal solutions. The ventilation of the zinc analysis room is taken care of by means of a small fan.

The balance room also adjoins the main laboratory. The slabs on which the balances rest are supported by concrete piers which do not con-

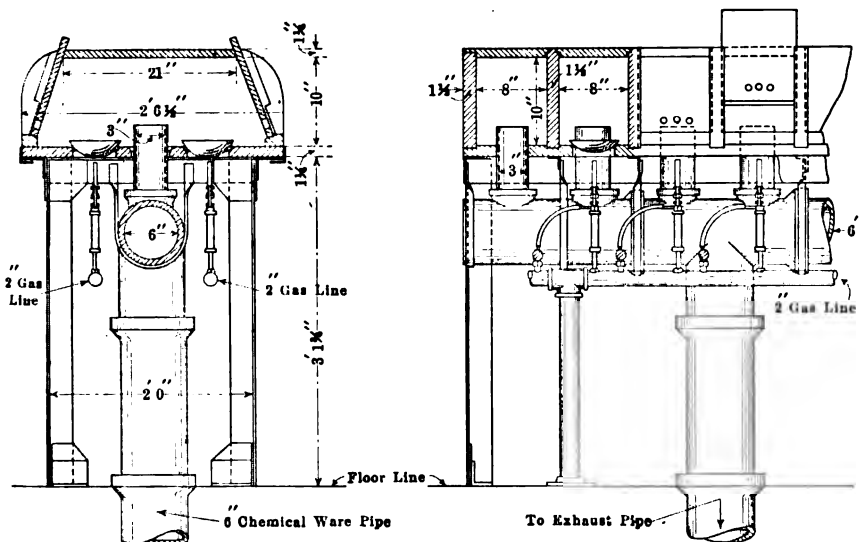


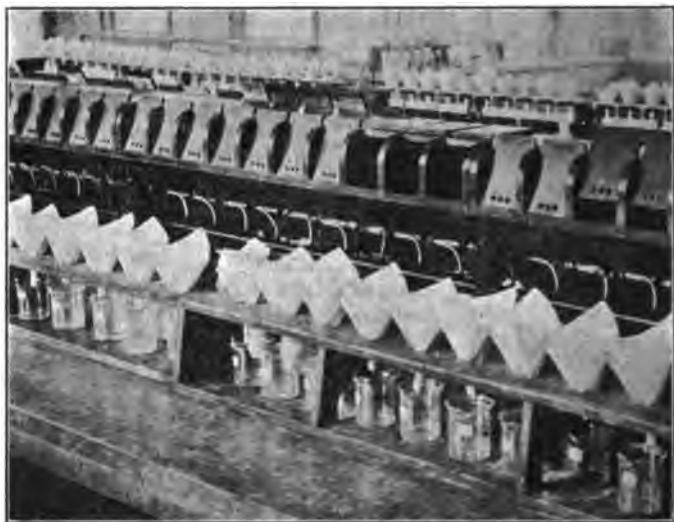
FIG. 3.—ALBERENE STONE HOODS.

nect with the building, and this removes annoyance due to jars. The room was built large enough to accommodate the filing case for the samples, which adds to the convenience of the work because the samples are readily accessible in case results are to be checked.

The experimental laboratory is provided with a hood somewhat similar to those in use in the Bureau of Standards. It is constructed of Alberene stone and wire glass. The top and sides being of glass makes the hood very light, and those who have worked in the old-fashioned wooden hoods can readily appreciate this advantage. The hood is downdraft, being exhausted by an aspirator similar to the one operating the hoods in the main laboratory.

The experimental laboratory is equipped to make any analysis that

may be required of the supplies furnished for the plant. Gas is used for heating purposes, in general, but electric hot plates or multiple-unit fur-



**FIG. 4.**

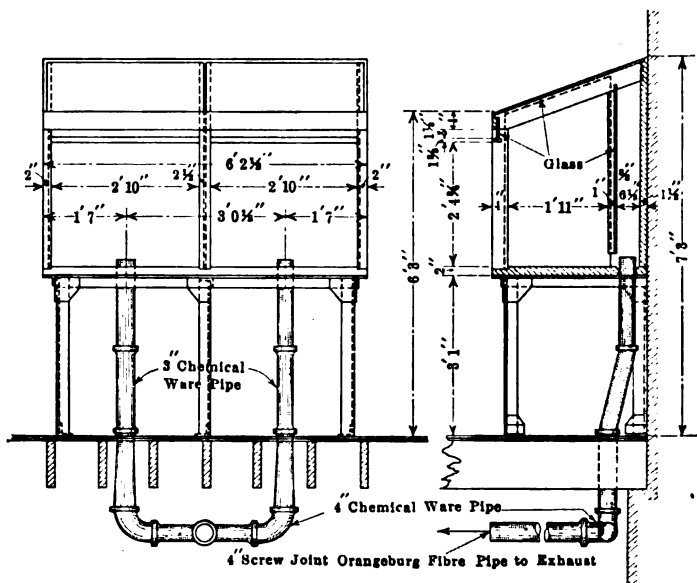


FIG. 5.—ALBERENE STONE AND GLASS HOODS.

naces are employed where it is possible to use them. For drying purposes the Freas electric ovens have been installed, and for electrolytic work a Veit apparatus has been provided.

As the supervision of the town water supply is entrusted to the Laboratory, it was found advisable to fit a room for carrying out both the chemical and bacteriological examination of the water. Our equipment is of the usual nature required for such work, consisting of a sterilizer, incubator, refrigerator and the other accessories.

This covers, in general, the principal features of our building, but before closing I would like to call attention to the size and location of our stock room, which is so situated that it is central to both the main and the experimental laboratories.

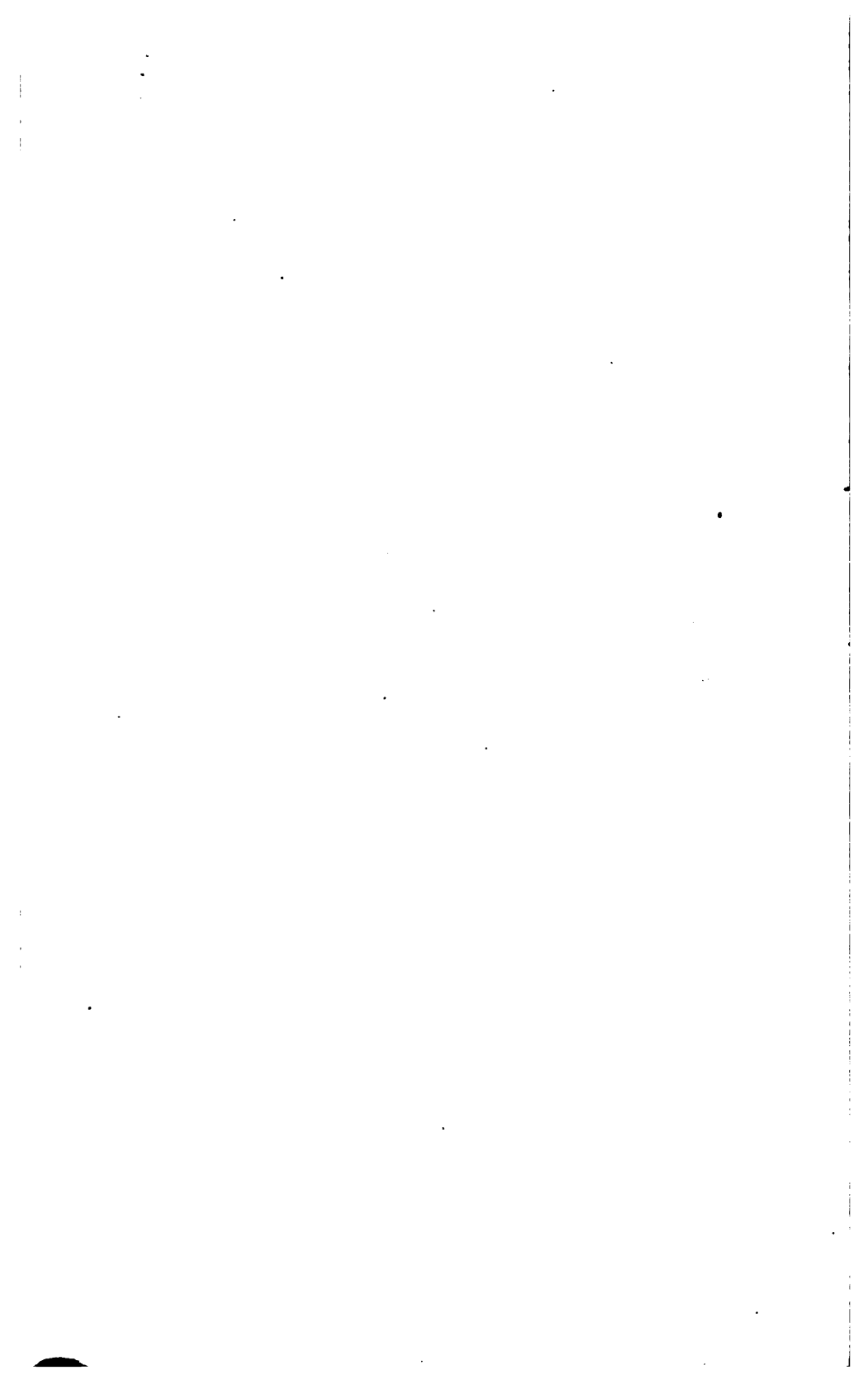
The library, also, is located so that it is easily accessible from both laboratories and is so furnished that the men can calculate and keep their records in this room and not have them distributed, as is the usual custom, throughout the whole building.

The rooms for drying and grinding of the samples are in the basement and are equipped with hot plates, bucking board and grinder. Only the small samples are handled in these rooms, as the sampling and grinding of the large samples is carried out at the sample houses situated near the different mills. The basement also affords ample room for storage and packing.

A Barnstead still of about 5 gal. capacity per hour furnishes us with distilled water which is distributed to all parts of the building by means of block tin pipes.

An adequate wash room is provided for the men, and individual lockers are furnished.

Although the laboratory was designed primarily for the analysis of zinc ores, we have found that it is capable of meeting any of the many problems that arise in a plant of this size.



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## The Enrichment and Segregation of Mill Tailings for Future Treatment\*

BY F. E. MARCY, † B. S., SALT LAKE CITY, UTAH

(St. Louis Meeting, October, 1917)

It is not my purpose to write a lengthy article or to attempt the solution of the problem I am presenting, but to call attention to what I believe an important issue, hoping that it may arouse in some one, or in some company, a sense of responsibility. I am referring particularly to modern up-to-date milling plants discharging tailings which under existing conditions cannot be further treated or from which a recovery cannot be made at a profit, but where a quantitative and qualitative analysis shows that a portion of these tailings might profitably be treated at some future time when our metallurgical knowledge has advanced, when we have cheaper power, chemicals, labor or other advantageous conditions.

### *Early Methods*

In our early milling methods very little attention was given to the care of tailings. It was thought that if the mill could be located by a stream or a river where the final disposal of the tailings could be forever lost sight of, it was highly desirable. We can now see and estimate what was lost in these rich tailings, not recoverable from the streams and the great rivers. Fortunately for many, the depositing of tailings in these streams and rivers was objected to by the farmers and finally prohibited by the State Legislatures.

In looking back over our past milling methods, it seems that the first endeavor was to make the maximum amount of money for a particular period, or year, regardless of the future, the depletion of the mine, or the conservation of the total mineral content. The dumping and sluicing of 3,000 tons per day, for a period of many years, of a copper tailing carrying

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\* Originally presented at a meeting of the Utah Section, on May 23, 1917.

† Consulting Engineer.

12 lb. of recoverable copper together with its portion of precious metals, the thousands of tons of valuable metal contained in the tailings that have been carried away by Silver Bow Creek, the streams of the Coeur d'Alene, the Flat River and Colorado districts, the enormous quantities of the fine slime tailings that were dumped into Torch and Linden Lakes, illustrate such extravagances.

Companies that have been compelled to impound their tailings for the sake of recovering their water are the ones where the best metallurgical work has been done. Where water has been cheap and abundant, too much has been used, which leads to dilution and poor concentrating work. Those milling companies that seemingly were not so favorably located, or the companies that were forced to discontinue the pollution of the streams, have found themselves in the possession of tailings that are of great value, due to the new methods of concentration and of recovery; of these, flotation has taken the foremost position. I am advised that the Canadian Government has taken the stand that the mineral wealth of the country belongs to the country, and the development of it is a privilege granted by the Crown; that the mineral resources of the Dominion cannot be squandered by loose and extravagant methods of extraction. We now have in this country definite laws preventing the scattering of waste products from mills, with the result that today millions of dollars are being recovered from these tailing dumps at a comparatively small expense.

The problem of future treatment of our present mill tailings is a much more difficult one for the future metallurgist and mill man on account of the recent great advancement in our milling methods; yet relatively the valuable content that still remains may have proportionately greater value as our mines become exhausted and our great mineral deposits have become definitely known and their value realized.

I do not share the view of those who consider that flotation is in its infancy, any more than the electrical industry is in its infancy. If so, they are infants of great proportion. In both industries we are fast approaching the maximum efficiencies. Yet the metallurgist will often find residual values from these modern concentrators and if they are not isolated by enrichment and segregation they will be forever lost on account of admixture with valueless material.

#### *An Illustration of Segregation*

I remember that while I was employed by the Anaconda Copper Mining Co., in 1901, this company made arrangements to segregate the fine slimes from the sands, supplementing the arrangement which their new plant afforded in saving all of the tailings instead of allowing a portion of them to go down Warm Springs Creek. It was realized, in starting this new concentrator, that these slimes were of great value on account of the copper content and, consequently, ponds were made, 300 ft. in width, 600



ft. in length and approximately 14 ft. in depth, to settle these values and decant the water. These ponds were operated intermittently—as soon as the ponds became loaded they were drained and then excavated by a drag line and bucket. The discharge from the bucket was piled along the side of the pond. Part of this slime was made into blast-furnace bricks and smelted, the remaining portion stored for future treatment.

A more detailed statement of the character of this particular material is found in an excellent paper by Ralph Hayden on The Concentration of Slimes at Anaconda.<sup>1</sup> The effect of saving these slimes for the last 16 years at the Washoe Reduction Plant has resulted already in the recovery of millions of pounds of copper with many millions more yet unrecovered. The present flotation slime plant at Anaconda which is working on these particular slimes recovers more than 1,000,000 lb. of copper per month, which is produced at a cost probably not exceeding 5c. per pound. The Chino Copper Co. segregated the tailings from its mill at Hurley and now has a retreatment plant handling 1,000 tons per day with further extension under construction. There are other companies that have segregated to some extent.

There are millions of tons of sulphide tailings that are now becoming slowly oxidized, and cannot be treated without installing expensive grinding machinery plus the flotation equipment; if the slime which carries the high values had been segregated, millions of pounds of copper could easily be recovered by the simple flotation methods of today, as illustrated by the Anaconda and Chino practice. While the sands, which are of lower value, would oxidize faster without the slimes and become available for leaching where coarse material is desirable.

The illustrations that I have given as to our past losses are in no sense given for the purpose of criticising our managers and millmen, for their achievements are all a matter of record that will go down in our metallurgical history; losses and wastes go hand in hand with new ventures and a new country. The thought I desire to set forth is our future responsibility in these matters.

#### *Kind of Tailings That May Be Segregated*

There are often concentrators in operation where some portion of their middlings cannot be brought to grade or a marketable product made. These should be stored separately and not mixed with the general tailings. We have not been able to recover the oxides and carbonates that are mixed with the sulphide tailings. J. M. Callow's filming experiments, which he has so carefully described,<sup>2</sup> point out what may be expected. There are millions of tons of tailings, however, whose oxide and carbonate content are even less than the tailings that are thrown

<sup>1</sup> *Trans.* (1913), 46, 239.

<sup>2</sup> *Bulletin* No. 122 (February, 1917), 245.

away by the present filming process, and it would seem that enrichment for future treatment should be made if possible. I think it is generally agreed that Mr. Callow is most familiar with the subject of the recovery of complex ores, and he has, at my request, made the following statement of his views as to what we may expect as to the probable recovery of such values.

"In the treatment of complex ores, such as those carrying lead, zinc and iron, and in copper ores containing both sulphides and oxides of this mineral, our present methods of recovery are still in embryo, and such tailings must inevitably carry an appreciable portion of the original contents of the ore, in the complex zinc, lead and iron ores especially. Sixty and 70 per cent. zinc recovery is often considered passable work, the rest of the zinc being either entangled or in some other way combined with the iron and lead which future inventions will no doubt show us how to recover. In the copper ores, flotation has added greatly to our past recoveries, but only in the recovery of sulphides; the oxides are still an unavailable asset, which in due time will undoubtedly be recovered by improved methods of extraction.

"In laying out a system for impounding tailings, the plan hitherto has been to impound them without any discrimination as to their values. In some ores it has been found that the principal values lost lie with the sands, and in others, with the slimes; so that a segregation might well be practiced."

As an illustration, let us assume the accumulative analyses of the tailings from one of the great concentrating mills where the process consists of tabling and flotation. The values given are in the form of copper contained in sulphides and carbonates. We will say that an extraction of 94 per cent. is made on the sulphides and 20 per cent. on the carbonates, making a total extraction of 82 per cent. The total copper remaining in the various screen sizes is given in Table 1.

TABLE 1.—*A General Tailing*

| Mesh | Per Cent.<br>Ind. Wgt. | Per Cent.<br>Cum. Wgt. | Sulphide<br>Cu Content | Oxide Cu<br>Content | Total Cu<br>Content |
|------|------------------------|------------------------|------------------------|---------------------|---------------------|
| + 48 | 1.6                    | 1.6                    | 0.009                  | 0.002               | 0.011               |
| + 65 | 16.2                   | 17.8                   | 0.030                  | 0.021               | 0.051               |
| +100 | 11.2                   | 29.0                   | 0.016                  | 0.012               | 0.028               |
| +150 | 11.0                   | 40.1                   | 0.019                  | 0.016               | 0.035               |
| +200 | 8.0                    | 48.2                   | 0.013                  | 0.013               | 0.026               |
| -200 | 52.0                   | ....                   | 0.051                  | 0.188               | 0.239               |
|      | 100.0                  |                        | 0.138                  | 0.252               | 0.390               |

On such tailings, it is possible with a Dorr classifier to separate the sands from the slime; the slime tailings may be divided again into two parts by deflocculating and decanting, making a fine crystalline mineral and a decanted slime. It is my belief that the decanted material will carry an enriched amount of oxide copper as compared with the general

mill tailings by two or three times. While it may not be possible with the present means to make a satisfactory and marketable concentrate, it would appear desirable to store separately rather than to mix with the impoverished tailings which might carry only 0.15 per cent. in copper, and hold such segregated material until metallurgy has advanced or the market is in condition to profitably treat it. I am not offering this as a solution to the problem of segregation of the tailings, but to point out what some preliminary experiments seem to show.

To retreat 15 or 20 millions of tons of unsegregated tailings in which a large portion carries only 0.12 to 0.15 per cent., is no easy problem for the future metallurgist.

There are many milling plants that treat complex ores which carry lead, zinc and copper, and it is not possible to make a satisfactory extraction from all of these values. In such plants, the extent to which enrichment and segregation of tailings can be carried on is limited only to the extent in which the expenditure seems desirable in the way of a future investment. Whether there ever will be a better opportunity for segregation, or greater ease of separation of these values, and whether it can be done at a lower cost than when the tailings were produced remains to be seen. Fresh and loosely combined material is always the most desirable to treat in all milling processes.

### *Storing and Importance of Milling Site*

In general, tailings that contain probable recoverable values should not be allowed to accumulate over large areas or in horizontal layers. Tailings from our first work in concentration generally contain the greatest values. Such tailings should be stored, if possible, so that those that are richest and most easily recoverable can be worked first, thus avoiding handling a large class of inert material when retreatment is commenced. It is certainly not desirable to pile the present low grade of tailings coming from mills where 90 to 95 per cent. recovery is made upon tailings from which only a 65 per cent. recovery was made.

I believe the segregation of tailings can be advantageously accomplished in many instances so that the weathering action will oxidize them for the ordinary leaching. The piling of tailings so that the maximum oxidation can take place, the elimination of all colloidal and talcy material would seem desirable, and a greater total recovery could be made from leaching and precipitation with iron.

The excellent results obtained at Anaconda in making building brick from segregated tailings is a subject worthy of consideration. It may surprise many to know the extent to which brick making may be successfully carried on to meet immediate market conditions, and the extent to which these brick may be shipped profitably to other markets.

*The Cost of Segregation*

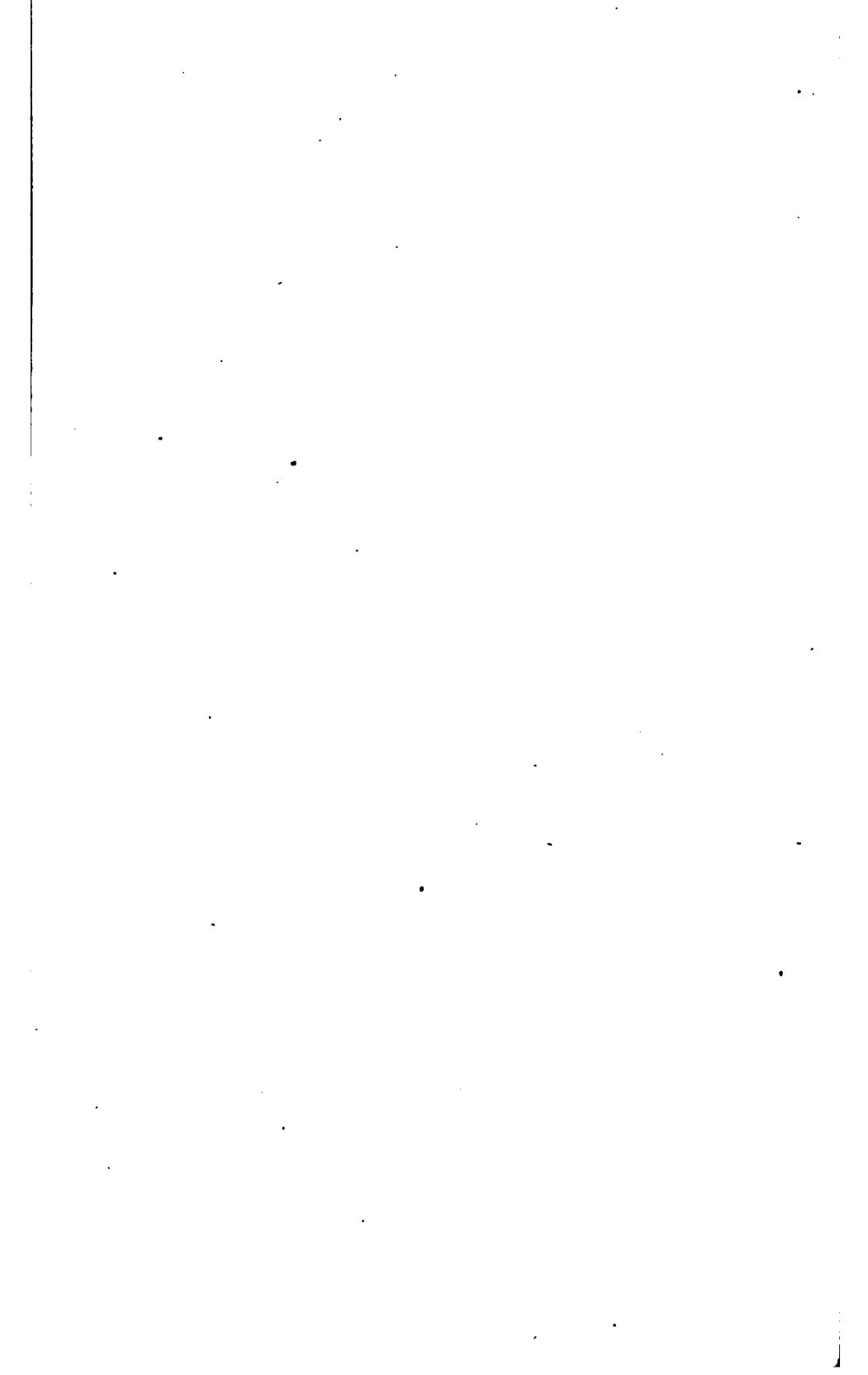
Segregation in general can be carried on, I believe, at a low cost, particularly at the time of the initial milling. The operation of tables, Callow tanks, Dorr thickeners, Cole drag-belt separators, excavating machinery and conveying equipment can be done for a very small amount per ton. There are many illustrations of the actual cost of stacking where mills are located on a flat site, as in northern Michigan and the zinc and lead districts of Missouri.

As to the charge for this work of segregation, it seems it is not a proper milling cost. For the production of this material credit might be given to extinguishment of ore reserves or an entirely separate account be kept. In my opinion, a most effective suggestion has been made by C. W. Van Law, who says: "To my mind this is not at all a charge to be carried against current operations, but is properly chargeable to capital as a deferred asset, just as in the case of the ore that is broken and carried in excess of requirements in shrinkage stope operations on a large scale. The cost of carrying the tailings away from the mill to dispose of them, so far as is necessary merely to enable continuous mill operation, is a proper current operating charge, but anything done in excess of this, where there is reasonable expectation of later obtaining more from the tailings than is at present possible, should be, in my judgment, carried on the books of capital account, their aggregate representing the value of the stored accumulation as an asset at exact cost of producing. Later, when these deposits are reworked, this asset should be scaled down, ton for ton as removed, corresponding reductions in the book asset being made at the same rate as they are created, to the entire extinction of the capital charge, such amortized asset being replaced by the aggregate profits realized from their reworking. I believe this to be an entirely logical and legitimate method of handling this matter, and it is what I should apply in my own case."

This is one of the problems of conservation of our resources and is analogous to many of the problems that have been so recently brought to our attention. The Government no longer allows the careless lumberman to set fire to the brush after he has removed the marketable timber, thus destroying the young trees and leaving the soil exposed and subject to erosion. The importance of a qualitative and quantitative study of these remaining values in tailings and the removal or relative enrichment is a subject for consideration and careful investigation.

If from our up-to-date milling plants there are millions of tons of copper-bearing mineral allowed to mix with worthless material, which when so mixed can never profitably be reworked, and if it is possible to segregate a considerable portion of this copper-bearing mineral so that it can profitably be retreated, then the responsibility of trying to save this large quantity of copper must rest with the administration and the board

of directors of such companies. It is not sufficient to turn this problem over to the operating mechanical superintendent who may lack metallurgical training, or operating metallurgists who likewise may be handicapped. Therefore, can we not suggest and point out the desirability of segregating and storing such tailings in a fashion that will best facilitate their future working and the saving of valuable metal?



### Tests on the Hardinge Conical Mill

Discussion of the paper of A. F. TAGGART, printed in *Bulletin* No. 124, April 1917, pp. 719 to 748.

JOHN W. BELL,\* Montreal, Quebec, Canada (communication to the Secretary†).—The test results in Mr. Taggart's paper will, I am sure, be recognized as a notable contribution, and of great assistance in the study of the performance of the Hardinge mill.

I regret, however, being obliged to note that Mr. Taggart still retains such confidence in the Kick-Stadler method of computing the "relative mechanical efficiency" of crushing machines. The Rittinger-Kick graph submitted by Mr. Gates<sup>1</sup> and the tests made at McGill University disclosed precisely the same fundamental defect in the Kick-Stadler theory.

Consequently, I have been obliged to recalculate in terms of Rittinger surface units the results obtained in the 28 rock-crushing tests cited by Mr. Taggart, in order to find out what the relative efficiencies really were. In order to show the large discrepancies between the Stadler and Rittinger R. M. E.'s, the most efficient result disclosed by each method is represented by the number 100, and the R. M. E.'s for the other tests have been recalculated on this basis. The results will be found in Table 1. Personally, I look forward to the time when we shall cease to talk about "relative mechanical efficiency" and merely refer to the "efficiency" of a crusher. All that is required to accomplish this is to agree on a standard method for determining the "crushing constant" of a given rock and a standard method for calculating the efficiency. The figures in the fourth column of Table 1 have been derived by assuming a constant of 2,000 for the quartzite crushed in the Yale tests.

The Stadler method sometimes indicates changes in efficiency produced by changes in operating conditions, as I pointed out in a paper describing rock-crushing tests made at McGill University. It is, however, not enough to determine that certain changes increase or decrease efficiency; surely it is equally important to determine the magnitude of these variations. By examination of Mr. Taggart's results, I have been obliged to conclude that the size of feed and amount of reduction greatly influence the Stadler R. M. E. figures, that they are positively misleading.

It should be mentioned that since the majority of the tests have been made at the least efficient feed rates, and since, moreover, it is one of the evidently very important factors affecting efficiency, it is possible that different results might be obtained by a high-tonnage feed series, and that

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† Received April 28, 1917.

<sup>1</sup> A. O. Gates, *Trans.* (1915), 52, 898, Fig. 20.

TABLE 1

| Test No. | Apparent<br>R. M. E. Stadler | R. M. E.<br>Rittinger | Efficiency Assuming Quartsite<br>Crushing Constant = 2,000 S. Units |
|----------|------------------------------|-----------------------|---|
| 202      | 21.4                         | 56.6                  |   |
| 203      | 23.7                         | 58.5                  |   |
| 204      | 21.0                         | 54.0                  |   |
| 205      | 25.4                         | 63.0                  |   |
| 206      | 19.9                         | 44.1                  |   |
| 207      | 21.4                         | 47.6                  |   |
| 208      | 26.3                         | 55.1                  |   |
| 209      | 18.6                         | 39.4                  | 10.5  |
| 210      | 16.7                         | 32.8                  | 8.7   |
| 211      | 18.8                         | 33.8                  | 9.0   |
| 212      | 21.4                         | 45.3                  | 12.0  |
| 213      | 22.6                         | 49.4                  | 13.1  |
| 214      | 22.2                         | 54.0                  | 14.3  |
| 215      | 21.6                         | 52.2                  | 13.9  |
| 216      | 39.8                         | 72.2                  | 19.2  |
| 217      | 67.1                         | 90.6                  | 24.0  |
| 218      | 94.5                         | 100.0                 | 26.5  |
| 219      | 100.0                        | 98.6                  | 26.2  |
| 220      | 19.4                         | 40.2                  | 10.7  |
| 221      | 20.8                         | 48.5                  | 12.9  |
| 222      | 23.5                         | 56.0                  |   |
| 223      | 24.6                         | 69.2                  |   |
| 224      | 22.6                         | 40.6                  | 10.8  |
| 225      | 24.6                         | 42.1                  | 11.2  |
| 226      | 16.4                         | 41.0                  | 10.9  |
| 227      | 13.5                         | 34.4                  | 9.1   |
| 228      | 12.3                         | 59.8                  | 15.7  |
| 229      | 4.3                          | 24.5                  |   |

some of the conclusions reached by Mr. Taggart or by me may require revision when this data has been obtained.

But even more striking than the errors in magnitude of Stadler R. M. E.'s, are the errors they lead to in some of Mr. Taggart's principal conclusions. He says (p. 734), "The R. M. E. of the machine increases with the feed rate up to 108 tons per 24 hr. beyond which we have an apparent condition of overloading." My conclusion is that the 144-ton feed rate (test No. 218) is the more efficient, and that there is consequently no indication of overloading. As an additional argument in favor of surface rather than energy units, I have plotted the results (see Fig. 1) of tests 213, 216, 217, 218, 219 given in Table 3 of Mr. Taggart's article, with the apparent Stadler R. M. E.'s and Rittinger R. M. E.'s figured on the same basis as in Table 1 of this discussion. That the Stadler R. M. E. should shoot up to a maximum value at 108 tons in a nearly straight line, and then down at 144, does not seem to me to be what one would reasonably expect. The Rittinger curve seems far more rational.



In regard to the result listed in Mr. Taggart's Table 4 (p. 741), I repeat a protest I have already made against the estimation of efficiency by considering the number of tons of -48-mesh material produced. The impression that is created by this table is, that a small feed will require 13 tube mills to produce 12,000 lb. per hour of -48-mesh material; whereas a high feed will require only four. On the assumptions made, numerically, this may be true, but a very practical consideration in this connection is that the 13-tube mill plant will produce a product containing 21.6 per cent. of -200 grade and the 4.3 tube mill plant product will only contain 4.7 per cent. of -200. Table 2 shows that by the addition of 1.7 tube mills, the amount of -200 could be nearly doubled with a

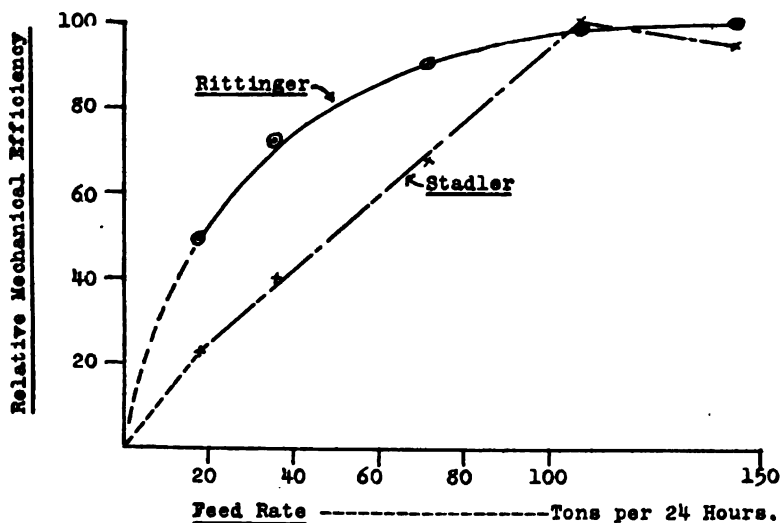


FIG. 1.

drop of only  $2\frac{1}{2}$  per cent. in the mechanical efficiency of the mills. The efficiencies are real (assuming the crushing constant to be 2,000) in order to eliminate the exaggeration of the effect of tonnage feed created by calculating R. M. E.'s.

TABLE 2

| Test No. | Tons Per 24 Hr. | Per Cent. -200 in Discharge | Efficiency |
|----------|-----------------|-----------------------------|------------|
| 213      | 18              | 21.6                        | 13.1       |
| 216      | 36              | 15.2                        | 19.2       |
| 217      | 72              | 8.4                         | 24.0       |
| 219      | 108             | 5.5                         | 26.2       |
| 218      | 144             | 4.7                         | 26.5       |

Speaking of his Table 4, Mr. Taggart says: "This table is based on

the assumption that the efficiency of reduction is the same on the smaller material returned to the mill as it is on the larger original feed." When it is considered that the Stadler R. M. E. figures cited by Mr. Taggart for a coarse-feed test and the finest-feed test are as 100 to 4, approximately, it is clear that the assumption is dangerous, and is even dangerous by the Rittinger theory which gives a ratio of 100:24½.

This naturally brings up the question of whether the oversize product from a ball mill should be returned to the ball mill or passed along to a second grinder (either ball or pebble mill) for final reduction.

On p. 745 Mr. Taggart compares tests 227 and 228 and says: "It is obvious, then, that the ball mill is a more efficient crushing machine than the pebble mill." My conclusion, arrived at by the Rittinger theory, is diametrically opposite, as will be noted by Table 3. That the pebble mill is much more efficient than the ball mill is well shown by the results in the last two columns.

TABLE 3.

| Test No. |         | Apparent R. M. E.<br>Stadler | R. M. E. Rittinger | Tons - 48-Mesh Material<br>Per Horsepower |
|----------|---------|------------------------------|--------------------|---|
| 227      | Balls   | 13.5                         | 34.4               | 0.38                                      |
| 228      | Pebbles | 12.3                         | 59.8               | 0.65                                      |

The fact that there was such a great difference in the feed diameter would render the comparison valueless or nearly so if the Rittinger R. M. E.'s had happened to be nearly equal. But since, in spite of this handicap, the pebble mill is able to demonstrate its great superiority, I am extremely doubtful of the advisability of returning anything but the very coarsest pieces in the oversize to the ball mill circuit, as it seems probable that the regrinding of the finer sizes could be done far more efficiently by a second pebble mill working in a closed circuit.

Mr. Taggart's conclusion "that the true maximum of the efficiency curve lies somewhere between 40 and 50 per cent. moisture" does not seem to be very well supported by his moisture-efficiency diagram, Fig. 8 (p. 732), since it would be, if anything, more reasonable to suppose that the maximum efficiency moisture was either 38½ per cent. or that it was somewhere between 25 and 40 per cent. The Rittinger results in Table 4 show that the maximum efficiency moisture will be found between 52 and 68 per cent. and is probably in the neighborhood of 55 per cent. It is worth noting, however, that the actual gain in efficiency realized by changing from a 25 to a 52 per cent. moisture would only amount to about 2½ per cent. of the power used (see 5th column, Table 1, tests 212-214). This of course applies to the tests cited. The increase might be appreciably greater for a large mill, fed at its maximum efficiency feed rate and feed size.

In regard to the efficiency effect of ball load, slope of mill, size of feed, dry versus wet crushing, etc., I do not think that very positive conclusions can be drawn because of the changes made in the mill adjustments before the required data was obtained. I am inclined to think that the trap crushes so much more easily than the quartzite, as to hardly warrant Mr. Taggart's first conclusion, which is based on tests 221 and 222. If the two rocks were similar, the R. M. E.'s would be the same. The Rittinger R. M. E.'s show an appreciably greater number of surface units produced per horsepower, and if we assume the crushing constant of the quartzite as 2,000, the trap constant (assuming that the mechanical efficiency of the mill was the same in each test) would be about 2,300. In test 223 an appreciably larger amount of work was done than in test 213 (see Table 4), and it is interesting to note that although the horsepower increased from 17.7 (in quartz test 213) to 20.1 (in the trap test) the R. M. E.'s are respectively 49.4 and 69.2.

If the mechanical efficiency of the mill was the same in these tests the trap constant would be raised to 2,800. It should be noted, however, that in tests 221-222, the ball load is 4,264 lb. (and the mill is level)

TABLE 4.

| Test No. | Rock Crushed | Ball Load, Pounds | Work Done per Unit, Surface Units. | R. M. E. |           |
|----------|--------------|-------------------|------------------------------------|----------|-----------|
|          |              |                   |                                    | Stadler  | Rittinger |
| 213      | Quartz       | 2,819             | 258                                | 22.6     | 49.4      |
| 223      | Trap         | 2,811             | 410                                | 24.6     | 69.2      |
| 224      | Quartz       | 1,406             | 183                                | 22.6     | 40.6      |

whereas in tests 213-223 it is only about 2,800 lb., with a mill slope of  $25\frac{1}{8}$  in. It is, of course, possible that the mill has a higher mechanical efficiency when grinding a softer rock, in which case the calculated constant 2,800 would be reduced.

The most efficient feed size is a matter of great practical importance. In regard to this, Mr. Taggart says: "The ball mill works more efficiently on material of intermediate (0.5 to 0.75 in average) size than on a coarser or finer feed" (Conclusion No. 3). There can be no question about the inefficiency of a ball mill working on a very fine feed, but I have the liveliest suspicions of the correctness of this statement in regard to the feed, coarser than the grade he fixes as most efficient. The coarse-feed tests (202 to 208) have noticeably high R. M. E.'s, but whether due to the coarse feed or the softer trap rock crushed it is difficult to say.

On p. 744, Mr. Taggart explains why he adopts conclusion No. 3. He points out that the Stadler R. M. E.'s show the coarser-feed test (208) to be more efficient than the finer-feed test No. 223, but he explains

that the probable reason for this is because the reduction ratio is only 7.0 in test 208 as against 45.6 in test 223. Consequently he says, "we may expect, by pushing the capacity (feed rate) in the case of the smaller (size of) feed until the reduction ratio is in the neighborhood of 7.0, that the relative mechanical efficiency will rise to 40, while from the same series of tests (the feed-rate tests) it is obvious that lessening the ratio beyond this point in the case of the coarser feed, by increasing the feed rate, would result in lowering the relative mechanical efficiency."

This reasoning is not at all obvious to me, because in my conception the ratio of reduction in the coarse test is more nearly 108 than 7, and in the finer feed test I would fix the ratio of reduction to be 14 instead of 46, and since these figures are diametrically opposed to his in direction, the conclusion to be drawn from them, following his own argument, is also reversed, that is, by increasing the coarse-feed rate, until the reduction ratio was reduced from 108 to 14, the coarse-feed R. M. E.'s. would go up by leaps and bounds as shown by the "feed-rate" tests. (See Fig. 1 of this discussion.) It is quite probable that the high R. M. E. in test 223 can be partially accounted for in this way.

The foregoing will make clear my reasons for believing that some of the numbered conclusions in Mr. Taggart's paper should either be reversed, or commented on, as follows:

1. The indications are that the trap crushes more easily than the quartzite, and that the efficiencies are therefore appreciably affected. The effect of small differences in rock constants is lessened by the fact that crushing machines utilize usefully a comparatively small amount of the power they draw.

4. A greater ratio of reduction in average size of material can be expected with coarse feed than with feed of intermediate size.

5. Pebbles working on a fine feed are much more efficient than balls working on a relatively much coarser feed, on account of the large reduction in the power required to lift equal volumes of pebbles (100 lb. per cubic foot) compared with balls weighing 250 lb. per cubic foot. The powers are indicated to be roughly proportional to the weights per cubic foot given. It is to be expected that if the size of feed to the pebble mill was gradually increased, a feed size would ultimately be reached which could be crushed more efficiently by a ball than by a pebble mill. These conclusions are based on tests 227-228.

12. The relative mechanical efficiency of the mill increases to the point of overload, which, however, was not reached in the tests described.

13. I hardly think Mr. Taggart has sufficient data to draw the conclusion he gives.

14. . . . "is in no way commensurate." NOTE.—Probably on account of the inefficiency of the chain drive.

17. The relative mechanical efficiency for the conditions prevailing

in the moisture series of tests, is at a maximum, when the moisture is at or slightly in excess of 52 per cent. of the weight of the pulp.

18. Probably dry crushing is less efficient than wet crushing, but the decrease does not appear to be very large.

22. It seems probable that the oversize from a ball mill could be more efficiently reduced in secondary mills using pebbles.

A. F. TAGGART, New Haven, Conn. (communication to the Secretary\*).—The writer wishes to record his appreciation of the careful study bestowed by Mr. Bell on the paper on Tests on the Hardinge Conical Mill and of the labor expended in translating the data therein contained into such shape as to make them comprehensible to those who use the "surface-unit" method of analyzing crushing data.

He wishes further, however, to register emphatic disagreement with the conclusions drawn by Mr. Bell and summarized at the end of his discussion. Most of Mr. Bell's conclusions are so completely at variance with the experience of practical mill men as to make repudiation here superfluous were it not for the fact that they were arrived at by applying a method of calculation ably defended by many writers on crushing data and therefore not to be lightly ignored.

To refer in detail to a few of Mr. Bell's criticisms: (a) As to the insufficiency of the data: The writer realized throughout the course of the experiments that rigorous proof of the conclusions drawn demanded more work than it was possible to do under the conditions that obtained. For that reason the data upon which the conclusions were based were fully presented in order that each reader might himself judge of their sufficiency. The writer further corresponded with and talked with several operators of mills before submitting the paper for publication, in order to determine whether or not the conclusions reached differed radically from mill experience, and was pleased to find remarkable agreement.

(b) Mr. Bell apparently overlooks the fact that the point of overloading in the operation of any crushing machine marks a sudden change in the phenomena involved. In some machines, such as rolls, stalling occurs; in ball mills there is a practical cessation of grinding, the mill acting as a conveyor only. Bearing this point in mind, the Stadler curve in Mr. Bell's Fig. 1 is more rational than the Rittinger curve, which latter would indicate a broad maximum and a gradual diminution in efficiency as the point of overload is passed.

(c) In regard to Table 4 (p. 741) the writer is far from defending the mill method of using "per cent. — 48 mesh" or any other mesh as a measure of crushing efficiency. However, such a means of measurement is used as a guide for practical work by intelligent operators of wide

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\* Received June 27, 1917.

experience and carries weight for that reason. The near agreement reached by its use with the conclusions of the writer is not the least argument in favor of the Stadler method of measurement.

(d) Relation between relative mechanical efficiency and metallurgical treatment: No attempt was made in the original paper to analyze the suitability of any particular product to subsequent mill operations. The requirements of these operations differ with every ore and every process. The effects of changes in the operating conditions on the relative mechanical efficiency of crushing are in no way changed by these other matters. Such analysis can be left to the mill manager.

(e) Effect of size of feed on the efficiency of reduction: In discussing the writer's Table 4, Mr. Bell says:

"When it is considered that the Stadler R. M. E. figures cited by Mr. Taggart for a coarse-feed test and the finest-feed test are as 100 is to 4 approximately, it is clear that the assumption is dangerous, and is even dangerous by the Rittinger theory which gives a ratio of 100 : 24½."

Mr. Bell has apparently compared tests 219 and 229, where practically the only similar conditions are that the tests were performed in the same laboratory with the same percentages of moisture, while for such comparison the only variable should be the size of feed.

(f) Pebbles vs. balls: It is practically the universal experience, where tests have been run in the mills, that the amount of grinding done in a ball mill per unit of power far exceeds that done in the pebble mill, and that it pays to install the additional power necessary and use balls instead of pebbles. Such a change has been made in many of the mills throughout the country. Mr. Bell notes that the pebble mill produces more "-48-mesh" material per horsepower expended than the ball mill. On the same basis of reasoning the tube mill is a far more efficient crusher than the gyratory, yet such is not the usual conclusion of mill men. In test 227 the feed was 9.900 mm. average size, in test 228 it was 1.173 mm. Obviously -0.295-mm. material can be produced with a smaller expenditure of power in test 228 than in test 227.

(g) The effect of moisture, point 213, Fig. 8, is obviously an accidental maximum, the position of which was determined by the moisture content in that particular test. It will be apparent to anyone accustomed to reading curves that such an accidental maximum might occur at any moisture percentage between 35 and 50. But it will be obvious to the same reader that the maximum of a smooth curve averaging the experimental points will lie between 40 and 50 per cent. and, as a matter of fact, very near 40 per cent. The conclusion drawn by Mr. Bell, that the point of maximum efficiency lies between 52 and 68 per cent. moisture, is utterly at variance with all mill experience where the question of most efficient moisture content has been tried out.

(h) Size of feed: The writer is unable to follow Mr. Bell's argument

under this heading, since the definition of ratio of reduction used by Mr. Bell is so widely divergent from the common definition, viz.:

$$\frac{\text{Average size of particle in feed}}{\text{Average size of particle in product}}$$

The fallacy of his method is proved by mill experience, which has taught operators to feed ball mills with a product in which a large percentage will pass a 1-in. ring, whenever the plant is of sufficient capacity to justify the installation of heavy rolls or disk crushers between the breakers and the ball mill.

(i) Conclusions:

1. The writer can see no reason from his data or Mr. Bell's analysis of the same to change his conclusion No. 1. There are unquestionably ores so hard and ores so soft that a comparison of the relative mechanical efficiencies of the conical ball mill working on two ores at the extremes of the list would show a marked difference, but for average ores the writer still believes that the character of the gangue has little effect on the relative mechanical efficiency of the mill.

4. As previously mentioned, Mr. Bell's definition of reduction ratio precludes discussion on this point.

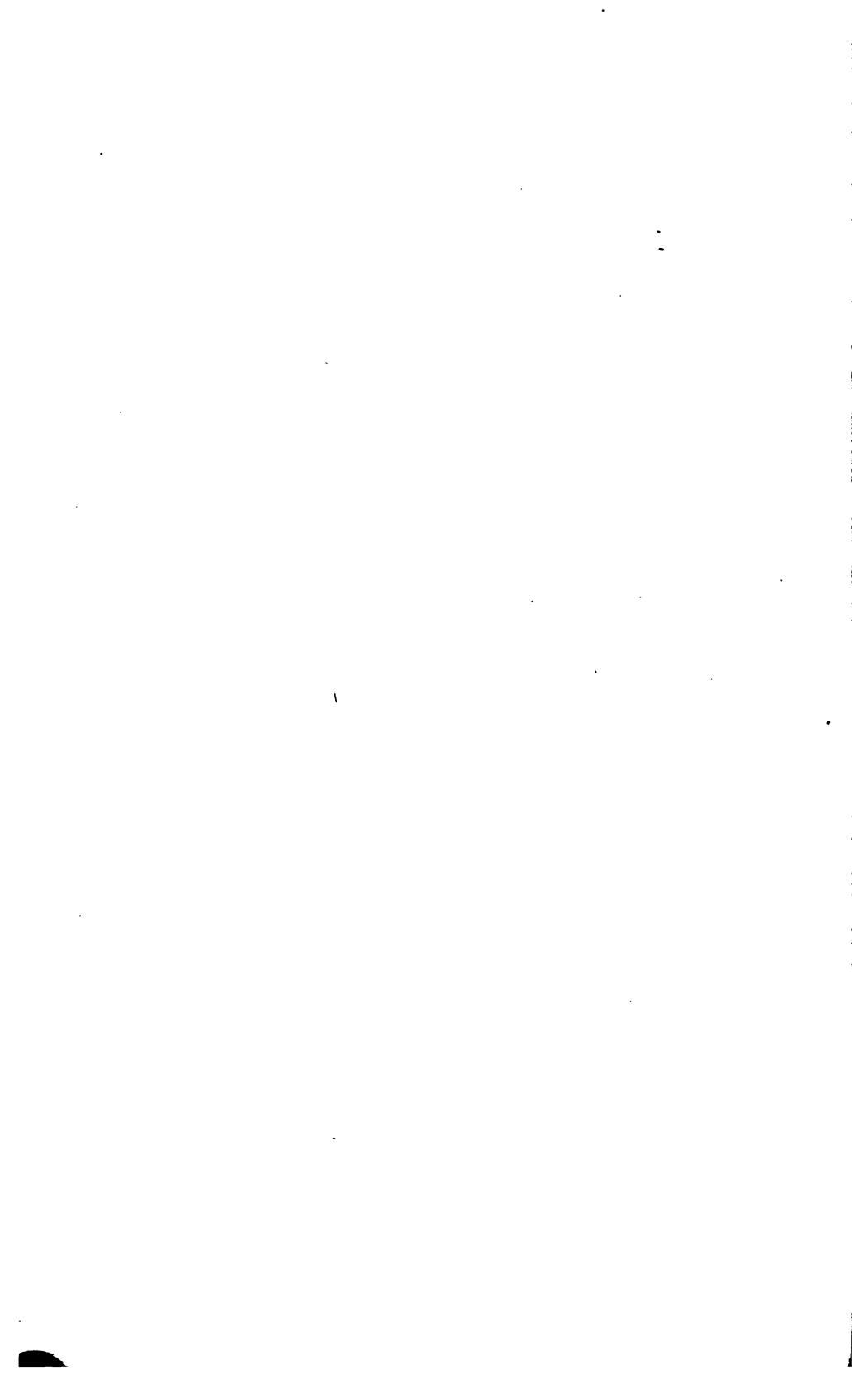
5. This conclusion in the original paper is almost unanimously supported by mill experience.

12. Mr. Bell is working under the disadvantage of not having seen the experiments and not visualizing accurately from the screen tests reported.

13. An operator of mills will have little trouble in agreeing with this conclusion.

14. Chain drive was not used on the mill in the Hammond Laboratory.

17, 18, and 22. The conclusions drawn here by Mr. Bell, using the "surface-unit" method of measurement, are the strongest arguments against the method that the writer has yet seen.





TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS  
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Salt in the Metallurgy of Lead\*

BY OLIVER C. RALSTON,† SALT LAKE CITY, UTAH, CLYDE E. WILLIAMS,‡ SALT LAKE CITY, UTAH, MARVIN J. UDY,§ MIDVALE, UTAH, AND G. J. HOLT,|| SALT LAKE CITY, UTAH

(St. Louis Meeting, October, 1917)

THIS paper reports the results of the use of salt in some research work carried on during the past 3 years at the Salt Lake City Station of the Bureau of Mines, which is quartered in the University of Utah, whose Metallurgical Research Department coöperates with the Bureau. Only the work involving the use of salt is described.

The authors wish equal credit given to J. F. Cullen, C. L. Larson, C. Y. Proutz, C. E. Sims, H. C. Neeld, R. W. Johnson, F. G. Moses, N. C. Christensen, and G. H. Wigton, who have all in some way contributed toward the success of this work.

### *Lead Metallurgy Popularly Supposed to be Without Problems*

In lead metallurgy it is difficult at first to find metallurgical problems, as the prevailing conception has been that lead is so easily concentrated in its various ores and so easily reduced to its metallic form that the only progress possible is that involved in cheapening methods of mining and of reduction. Most of the problems usually considered are of minor nature and do not involve radical metallurgical changes.

During the 3 years of work of the Salt Lake Station, the following classification of problems in the metallurgy of lead has been made:

Those involving

#### *Oxidized Ores.*

1. Ores containing only lead.
2. Ores containing lead and silver or gold.
3. Ores containing lead and zinc, with or without silver.

#### *Sulphide Ores.*

1. Ores of pyrite carrying some lead.
2. Ores of complex zinc-lead-iron sulphides.
3. Complex sulphides containing in addition precious metals.

\* By permission of the Director, U. S. Bureau of Mines.

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The problems involved in the treatment of oxidized ores of lead are due partly to the nature of the ores themselves and partly to their physiographic environment. It is well known that many lead-carbonate ores cannot be concentrated successfully because of the large losses of lead carbonate in the slime. A circular was issued by the Bureau of Mines in May, 1916, on The Flotation of Oxidized Ores, by O. C. Ralston and G. L. Allen, which described work in the flotation of such slimed-lead carbonate, and there are now a number of mills in operation utilizing the method. This method involves the use of the principle of sulphidizing the lead-carbonate particles by the addition of a soluble sulphide, as *e.g.*, sodium sulphide, sodium polysulphide, calcium sulphide, etc. Many oxidized ores contain compounds of manganese or basic sulphates of iron, or peroxides of lead which react with the soluble sulphide used and do not permit the sulphidizing of the lead carbonate particles; as a result, this method cannot be employed for such ores, and the slime problem still exists. In other cases, low-grade oxidized ores of lead are found in localities distant from water so that wet concentration of these ores is impossible. Dry gravity concentration is subject to dust losses in the same way that wet concentration is subject to slime losses, and has the further disadvantage of lack of recognition as a standard metallurgical method.

There are portions of Nevada, Arizona, California, Utah, and other Western States far removed from sources of fuel so that only a self-fluxing ore can be smelted at a profit in case the ore is not capable of concentration. In these States it is not uncommon to find mines situated on railroads and not far distant from smelters, whose siliceous lead-carbonate ores are not mined unless the metallic content amounts in gross to from \$8 to \$14 per ton. The Tintic district, 100 miles south of Salt Lake City, is an example of this. The tonnage of this type of ore which has been developed in many of the older Western mining districts is often large.

The same remarks apply to the other types of lead ores above mentioned. Throughout the semi-arid Western mining States are mines in which the water level is quite deep and in which large amounts of these various "dry" lead or lead-silver ores exist.

The presence of small amounts of galena intimately crystallized with pyrite is also a common low-grade type of ore which is used to some extent for the fluxing value of the iron but which is penalized with a heavy roasting charge on account of the high sulphur content. Where the lead cannot be liberated by grinding, this ore has never been milled and is often left untouched in the mine. The southeastern Missouri mills also produce a middling of this description which is largely stored or lost, or milled for the recoverable lead and discarded with some loss.

The problem of the treatment of complex sulphides is so well known that it hardly needs description. The presence of zinc in these mixtures has usually been the stumbling block. The present methods of making

electrolytic zinc or of making leaded-zinc-white, have tended toward solving these problems, although the results are still unsatisfactory either on account of the high installation and operation costs or on account of low extractions. The presence of precious metals adds a complicating factor.

### *Leaching with Salt Suggested by Salt Lake*

The use of salt as a metallurgical reagent was immediately suggested by the presence of the Great Salt Lake and by the past work that has been done in "chloridizing" ores with salt. There is not much in the literature about the effects of salt on the lead content of the ores treated by the old chloridizing methods, as lead was not worth much in the days when those methods were applied. In the hyposulphite lixiviation processes, it was known that some lead dissolved in the "hypo" solutions, but the effect of salt during roasting was largely neglected. Nevertheless, as long ago as 1854, M. Becquerel, of Paris, reported work which he had done in roasting lead-sulphide ores with salt, followed by a brine leach in which silver chloride and lead sulphate dissolved, and from which he proposed to recover metallic lead by electrolysis.<sup>1</sup> Due to the fact that there was no commercial electric generator available at that time, his process was doomed to failure.

Chloridizing mills were in operation at Park City and at Silver City, Utah, by Holt & Dern and Knight & Christensen, respectively, at the time that our work was begun. In each of these mills a silver-gold-copper ore was given a chloridizing blast-roast followed by a brine leach, and in each case a small amount of the lead in the ore was found to go into solution, although it was incompletely precipitated by the scrap iron which was used to remove silver, gold and copper from the solutions. The brines in these two mills were also too dilute to allow of efficient work in the extraction of lead.

The Bunker Hill & Sullivan mine, at Kellogg, Idaho, was also the scene of some interesting work involving a hydrometallurgy of lead, as the Malm dry-chloridizing process was being tested out at the time this work was started. The process involved the treatment of the ore with cold chlorine gas to form chlorides of zinc, lead, iron, copper, silver, and gold, followed by a roast to break up the iron chlorides, and a brine leach to remove the soluble chlorides. The method depended for its success upon the use of zinc compounds to precipitate all of the other valuable metals from solutions with subsequent electrolytic recovery and re-use of part of the zinc. One reason that the process failed on the Bunker Hill & Sullivan ore was the low zinc content of the ore.

Standing on the shoulders of accumulated knowledge we utilized old facts for new purposes. The processes here described were gradually

<sup>1</sup> *Comptes rendus* (1854), **38**, 1095.

worked out, not in the exact order of their description, but in stages, following out various ramifications at various times.

### *Treatment of Oxidized Ores*

The ores containing only oxidized forms of lead and no other valuable metals have been treated successfully by several different methods in our laboratory. One of the simplest methods involves leaching the ore with a strong sodium chloride brine acidified with sulphuric acid. This method depends upon the fact that lead sulphate and lead chloride are both soluble to a considerable extent in a saturated salt solution. "In the cold" it is possible to prepare solutions containing as much as 1.5 per cent. Pb, although it is safer to use enough brine to make solutions containing 1 per cent. Pb or less. If the solutions are heated to the boiling point, it is possible to get concentrations of lead as high as 8 per cent. from lead sulphate and 12 per cent. from lead chloride. Most of our work has been done with cold solutions. After the lead is leached from the ore, it is precipitated from the solutions by the addition of lime, or by electrolysis, depending upon which method is to be preferred.

Several series of small-scale cyclic leaches with both types of precipitation have been made in our laboratory, in order to gain an idea of the impurities which tend to build up in the solutions and their effect on the efficiency of the process. Most of the lead carbonate ores are easily leached and have given extractions of from 80 to 98 per cent. of the total lead, depending upon what portion of the lead was present as unaltered sulphide. The much higher extractions possible with a hydrometallurgical process, as compared with a gravity concentration process, make leaching desirable.

In precipitation with lime, some calcium carbonate is thrown out of the solution together with the basic hydroxide of lead, and if ferric salts or other metal salts are present they are also liable to contaminate the precipitate. It was found that pre-treatment of the solution with powdered lime-rock precipitated the iron and alumina, formed some calcium chloride which precipitated all of the sulphate present, and did not precipitate the lead. This purified the solution to such an extent that the grade of the lime precipitate could be raised from 30 per cent. Pb to 70 per cent. Pb. In some of the cyclic work, careful washing of all of the salt from the precipitate allowed the preparation of a basic lead hydroxide precipitate containing over 77 per cent. Pb. In all cases this precipitate is somewhat discolored by a small amount of manganese leached from the ore and precipitated with the lead. This precipitation may be prevented by oxidizing the manganese to the manganate or permanganate stage, since these compounds of lime are extremely soluble. As a rule, there is no reason for doing this because the precipitated lead hydrate has never

shown any value as a pigment and our efforts to date have been to produce a high-grade lead product capable of easy reduction to the metallic state.

The lime precipitation is so simple, and settling and filtering of the precipitate so easy, that there is little more to be desired. In case most of the salt is not washed out, a high percentage volatilization of the lead takes place during reduction of the precipitate, the volatile compound being lead chloride. Only in ores that contain some ferrous compounds like siderite, and which hence produce reduced solutions of iron, is there any difficulty in obtaining sufficient purification of solutions with lime-rock preceding precipitation with milk of lime. Under these conditions, long aëration with compressed air in the presence of manganese compounds is necessary during the purification in order to oxidize ferrous hydroxide to the less soluble ferric hydroxide.

Electrolytic precipitation of the lead from the brines by the use of insoluble anodes was not successful, due to re-solution of the deposited lead by chlorine dissolved in the brine. Only iron anodes were found to be permissible. This resulted in a high energy efficiency of deposition, as much as 17 lb. of lead being deposited by the use of 1 kw.-hr. and the solution of one chemical equivalent of iron at the anode. Roughly,  $\frac{1}{4}$  lb. of iron goes into solution for every pound of lead deposited at the cathode. The voltage necessary for these electrolytic conditions is from 0.3 to 0.8 volts and current densities as high as 60 amp. per square foot are permissible.<sup>2</sup>

The lead is deposited in the form of a sponge which sinks to the bottoms of the electrolytic tanks and from which it can be gathered, washed, dried or pressed, and melted to bullion. The solutions do not build up in iron very much but occasionally excess iron will probably have to be precipitated with lime, a step which has been tested on a relatively large scale at Kellogg.

The necessity of using iron anodes, while it makes a very small energy consumption, is the main drawback to this process of precipitation. Although the electrolytic equipment would be small, it would be more expensive than the simple tanks and filters necessary for lime precipitation and the only place where electrolytic precipitation would be of any advantage would be in the case of ores that give foul solutions of manganese or in which notable amounts of oxidized zinc which passes into solution with the lead, are present. In this case, the lead can be precipitated by electrolytic methods and the solutions later purified and the zinc precipitated. This phase of the subject has not yet been tested by us beyond the determination that a saturated salt solution will not dissolve much

<sup>2</sup> See paper by C. E. Sims and O. C. Ralston: *Transactions, American Electrochemical Society*, Sept. 28 meeting, 1916, New York City, The Electrolytic Recovery of Lead From Brine Leaches, also *Metallurgical & Chemical Engineering* (1916), **15**, 410.

zinc chloride. This might be expected from an inspection of the data on the solubility of zinc chloride in solutions of sodium chloride.

Summarizing, the process developed consists in leaching a lead-carbonate ore with a saturated sodium-chloride solution, acidified with sulphuric acid, and used in such a ratio to the ore that the solutions shall contain not more than 1 per cent. Pb. After about  $\frac{1}{2}$  hr. of agitation, the solution of the lead is complete and lime-rock is added for about 10 min. agitation in order to purify the solution of iron, aluminum and sulphates, and the solution is then filtered from the gangue. Precipitation of the lead with lime leaves a brine which can be used over again. Electrolytic precipitation, using soluble iron anodes, is an alternative.

Commenting on the above process, it must be said that cheap salt, sulphuric acid and lime are necessary. Heretofore there has not been a cheap sulphuric-acid supply in the intermountain regions, but this condition is fast giving way to a new condition in which a number of the smelting companies are anxious to find a market for large amounts of acid. Salt must be cheap, because some is lost in the tailing. Not much washing of the tailing is allowable, as the leaching solutions must be kept saturated in order to dissolve lead. On that account, tailing containing as much as 50 per cent. solution or 12.5 per cent. salt will occasionally be formed, and the only wash water that can be used is the equivalent to that which is held by the damp tailing as compared with the original ore. Not more than two-thirds of the salt can be washed out with this small amount of wash water and the final tailing could hence contain as much as 4 per cent. NaCl, although it is usually less. Since salt can be marketed in Salt Lake City for \$2 per ton, and the cost of production of Kansas, New York, California and Utah is known to average less than \$1.50 per ton, it is almost certain that there are many localities in which cheap salt will be available. Finally, lime-rock is available in practically every mining locality and can be burned to calcium oxide for about \$1.50 per ton and marketed in many localities for not more than \$4 per ton.

The second method developed for application to oxidized lead ores was an igneous one in which the temperatures involved did not approach the melting point of the gangue. By mixing salt with the ore and heating to 850° C., it was found that all of the lead would volatilize as lead chloride, without regard to the composition of the gangue. In case powdered coal was mixed with the lead-carbonate ore and the mass moistened slightly so that it left a fluffy mass on drying out, it could be blast-roasted with as low as 2 per cent. of fuel, provided the ore was finely ground. The blast of gas through the charge assisted in the evaporation of the lead chloride and a 3-in. bed would be finished in about 15 min. after ignition. The white fumes of lead chloride contained a portion of the lead in a form which dropped in the flues a short distance from the downdraft blast roaster. The remainder was caught in a single-pipe Cottrell precipitator

which consisted of a 4-in. pipe 5 ft. long standing vertically, in the center of which hung an insulated wire connected to the high-tension pulsating unidirectional current from a transformer and rotary switch, the precipitator being operated in the standard manner. Five-inch pipes are in fairly large-scale use at the present time and the precipitator might well have been a 5-in. pipe instead of 4-in., but as it was it represented commercial conditions fairly well for the purposes of testing the possibility of precipitation of lead-chloride fumes. All visible fume was removed when gas velocities ranged from 4 ft. to 19 ft. per second, and sodium-sulphide paper held for 15 min. in the exit gases failed to show perceptible blackening when gas velocities of not over 10 ft. per second were used. The temperatures of the gases in the small precipitator varied widely, but were usually close to 100° C.

The use of an ordinary reverberatory roasting furnace in place of a blast roaster for carrying out the first step of the volatilization process is permissible. It has been found that the mixture of salt and ore in passing through such a furnace gives perhaps even higher extractions, although the cost of such a roast is higher, due to higher fuel consumption and greater length of time during which the ore is in the furnace. With a downdraft roaster, such as the Dwight-Lloyd sintering machine, the ore is in the roaster only about 15 min., so that large capacities are obtained in relatively small floor space and with relatively low first cost. As a substitute for the Cottrell precipitator, the bag house is acceptable, although for the same capacity the installation of a bag house costs more than a Cottrell precipitator and the upkeep of bags is a greater expense than the upkeep of the pipes or plates of the electric precipitation method.

Comparing the volatilization method with the leaching method above described, and with ordinary smelting methods for low-grade lead ores, it can be seen that the volatilization process has many great advantages. It is independent of the composition of the gangue, and non-fluxing ores are to be preferred on account of the fact that they do not tend to fuse in the furnaces. The hydro-metallurgical process is often hampered by the formation of slimes during leaching. This and the numerous other woes of hydro-metallurgy are all avoided by the volatilization process. Figuratively speaking, the volatilization process may be regarded as one in which gases are used for leaching ores. Gases are much more cheaply pumped and filtered than are solutions. The Cottrell precipitator corresponds to electrolytic precipitation from leach solutions, while the bag house corresponds to an ordinary filter. The downdraft roaster corresponds to the leaching tank.-

We have considered the evolution of this process as one of the most promising steps made in the development of a new metallurgy of lead.

The questions as to the further treating of the lead-chloride fume, which might have proved very difficult, fortunately gave us no trouble.

It was found that the method of handling copper chloride, which is now in use at Chuquicamata, was applicable to the lead-chloride fume from the Cottrell precipitator. Upon mixing the lead chloride with between 5 and 10 per cent. of coal fuel and its chemical equivalent of lime and heating to a dull red heat in a reducing atmosphere, lead chloride is easily transformed into metallic lead with the formation of a calcium-chloride slag. The calcium chloride was found to be a suitable substitute for sodium chloride in the first step of the process, as the silicate of calcium is less fusible than that of sodium, thus decreasing the tendency toward sintering. Thus it would be possible to recover somewhat over 50 per cent. of the chlorine for the first step of the process. Careful tests have shown that from 105 to 150 per cent. of the theoretical NaCl or mixture of NaCl and  $\text{CaCl}_2$  is necessary during the volatilization step of the process.

This process promises to be one of the most powerful processes on account of the wide range of ores to which it can be applied and on account of the high extractions of lead possible. Particularly, as compared to concentration followed by smelting, it gives a much higher percentage of extraction in fewer steps.

#### *Oxidized Ores of Lead Containing Precious Metals*

In case silver is present in oxidized lead ore, it is very often in the form of silver chloride or silver chloro-bromide. Both lead and silver chlorides are soluble in strong brines, so that the leaching process for lead-carbonate ores can also be applied to some of the ores containing silver. The silver sulphides dissolve so slowly in an acidified brine that the method is not applicable to them. Leaching of such ores depends upon the mineralogical association of the silver minerals. Most of the gold which is often found in such ores is likewise insoluble.

It was found that chloridizing roasting and brine leaching of lead ores containing precious metals could be accomplished, although considerable difficulty was met in the extraction of the lead due to the formation of a lead silicate or other compound which is insoluble in brine. Most of the gold and silver could be extracted after a chloridizing roast. However, it was found more desirable to raise the temperature so that from 70 to 90 per cent. of the lead volatilized as chloride leaving gold and silver chlorides in the ore to be leached with a brine. Some of the gold and silver was volatilized with the lead.

While this process was found to be successful and to promise commercial success as well, it was found that on still further raising the temperature it was possible to volatilize all of the lead, gold, and silver. At 900° C., in a reverberatory furnace, giving the ore at least an hour's roast, it was possible to volatilize out the chlorides of lead, gold, and silver



to such an extent that only a trace of lead was left in the calcine and 10 to 20 per cent. of the gold and silver. In other words, the extractions were 99 per cent. of the lead, and 80 to 90 per cent. of the gold and silver.

In no case have such ores been found to carry more than 10 per cent. of lead. These are known as "dry ores" in lead smelting, as at least that much lead is desirable to allow good extraction of gold and silver. The volatilization process seems to be especially adapted to this type of ore. Since no complex machinery is involved, it would seem that small inexpensive units can easily be erected to treat such low-grade ores economically. It is known that many mines in the arid and semi-arid regions need a process of this description.

Occasionally some copper is present in such ores, and it has been found that extractions approximately as high as those of lead are possible. Zinc, on the other hand, does not yield to this process, the volatilization varying widely for different ores, and being entirely uncontrollable. Some ores have given a maximum volatilization of 75 per cent. while others have given a minimum of 5 per cent., but in no case has it been possible for us to control or predict the amount. A mixture of gold, silver, lead, and copper in an ore is not a hard thing to handle by present standard processes, but the presence of zinc has long been a problem in the lead and copper smelters of the intermountain States. This new process seems to be much better adapted to treating such ores than the present smelting processes.

This process is now being tested on its siliceous oxidized lead-silver ore in a semi-commercial plant by the Chief Consolidated Mining Co. of Eureka, Utah. Other installations are contemplated on a larger scale. Some similar work is now in progress in our own laboratory in an attempt to determine what form of furnace can be used to best advantage.

The only limitation to this process is that the ores treated must usually be low in sulphur and not too near to being self-fluxing. From 2 to 3 per cent. sulphur present in the ore usually gives low silver extraction. It is possible that pre-roasting of sulphide ore followed by the addition of salt will improve this condition, although a few roasts of sulphide ores carried on in our laboratory to test this idea have not given good extractions. However, for the low-grade oxidized ore problem which was met in the intermountain States this process seems to be a solution.

### *Oxidized Lead-Zinc Ores*

Most of the oxidized ores containing zinc, when leached with sulphuric acid form enough silicic acid to prevent filtration of the solution. To overcome this, it was found best to apply strong sulphuric acid in sufficient amount to form sulphates of zinc and lead, the mixture of ore and

acid forming a thick mud. On passing this mud through a furnace at about 600° C., all silicic acid is dehydrated and all iron sulphates are broken up, leaving only zinc sulphate soluble in water. The lead sulphate formed is left in the residue from the water leaching of zinc sulphate and can then be removed with a strong brine, although lead sulphate which has been heated to the point of formation of basic lead sulphate is very hard to extract with a brine or an acid brine.

A better method of handling such an ore seems to be to use the volatilization method which has been mentioned above, since heating the mixture of salt and ore causes volatilization of lead chloride, usually leaving most of the zinc untouched. The zinc can then be leached by ordinary methods. By conducting the volatilization at over 900° C. for nearly 2 hr., it was found that only minute traces of chlorine were left in the calcine, as all the chlorides volatilized. Sulphuric-acid leaching and electrical precipitation of the zinc from the solution is possible, although chlorine is known to be an element undesirable for electrolytic precipitation of sulphate solutions.

In case a large amount of lime or similar gangue is present in an ore of lead and zinc carbonate, it is perhaps best to treat it by the standard igneous method for making of a leaded zinc-oxide pigment. The plant involved in such work is rather small and need not be expensive, but since such a process does not use salt it need not be discussed in this paper.

### *Sulphide Ores*

For the treatment of sulphide ores containing lead, the use of a salt metallurgy would also seem to present some advantages. In the case of low-grade galena ores needing concentration before smelting, the methods of the past have often been wasteful, as concentration usually did not give high recoveries. Most of the losses were in the slimes and have now been corrected by the use of flotation, although extractions are often still far from satisfactory. The volatilization method mentioned for oxidized ores can be used on these ores, and its great advantage is that it is possible to produce metallic lead as finished product when treating a low-grade ore and at the same time make a much higher extraction than was possible by the combined concentration, flotation and smelting methods. This high extraction is to be balanced against somewhat higher costs.

It has been found possible to roast such ores at lower temperatures and form lead sulphate which is soluble in saturated brine, and with the presence of some salt the roasting is facilitated so that it may be carried on at still lower temperatures. Very little lead is volatilized and all gold and silver present can be chloridized so that the lead, gold and silver can be leached with strong brine. This contains the basis of the leaching

process which has recently been tested on a semi-commercial scale by the Bunker Hill & Sullivan Co. at Kellogg, Idaho, and is still under consideration by them.

In the southeastern Missouri district there are always some iron middlings formed in the milling of the lead-sulphide ores. All the iron middlings contain 10 to 15 per cent. of lead and occasionally some zinc. A mixture of salt with this material, passed over a downdraft roaster with the shallow bed and a low draft allows the volatilization of a fume of lead chloride mixed with elemental sulphur from the pyrite. The pyrite acts as the fuel and continues to burn after the lead chloride has been volatilized, but it is not necessary thus to burn up all of the pyrite. It is possible in a 15-min. treatment to send off most of the lead and leave a partially roasted pyrite which can be discharged from the downdraft roaster while still ignited, and quenched with water. No work has as yet been done with the mixed sulphur-lead chloride fume to see if it can be successfully handled. If zinc is present as zinc sulphide, there is not so much danger of overheating the mass with resulting matting of the iron sulphide, and good extractions of the lead are thus possible. Under no condition has it been found possible to volatilize out any silver present in mixed sulphide ores. The zinc is unaffected by the roast if it is stopped as soon as the lead is volatilized. It has occasionally been possible to grind the quenched calcine for flotation of the zinc sulphide to form a concentrate containing 40 to 45 per cent. zinc. The proper roast that will allow this is rather too difficult at present for commercial application, but the idea is a good one with which further work might be done.

For the treatment of complex sulphides of zinc, lead, iron, copper, silver, and gold, a salt metallurgy does not seem to present a solution. It is possible to roast such ores with salt and save at least part or all the lead, leaving the other metals in the residue, but it can be seen that this does not constitute a solution of the problem, and bears out our statements that the large field of application of these processes comes in the treatment of oxidized ores.

Summarizing, we may say: 1. For the treatment of oxidized ores of lead two methods are available. One involves the leaching of the lead with a saturated salt solution acidified with sulphuric acid, followed by either electrolytic precipitation of metallic lead, or lime precipitation of the basic lead hydroxide with the concomitant regeneration of the salt solution. The other method involves mixing salt with the ore and treating in a blast roaster, or a reverberatory furnace, at a temperature of 800° C., or more. At these temperatures lead-chloride fumes are formed which can be caught in a Cottrell precipitator, and later reduced to the metallic form by treatment with lime and carbon in either a melting pot or a small reverberatory furnace, with the formation of calcium-chloride slag which can be used as a substitute for sodium chloride in the volatiliza-

tion step of the process, and means the recovery of from 50 to 70 per cent. of the chlorine.

2. For oxidized lead ores containing precious metals or copper, the leaching process can be used after a chloridizing roast at slightly over 600° C. The volatilization process can be used provided temperatures of over 900° C. are used, the ore does not tend to form a slag, and the percentage of sulphur in the ore is low. The zinc in most ores is only slightly affected by this latter process.

3. Lead can be recovered from practically any sulphide ore by the volatilization method, but precious metals and copper cannot be extracted. Precious metals and copper are leached after a chloridizing roast, although the roast is more difficult of accomplishment than is the case with oxidized ores.

4. The volatilization process promises the solution of the vexing oxidized siliceous lead ores problem so common in the Western arid regions and has the further advantage of requiring very little water for the process. The chloridizing roasting and leaching method promises close competition to treatment of low-grade sulphide ores containing precious metals by gravity and flotation concentration followed by smelting, because of the use of only a few simple operations, the production of finished metal, and of the higher total extractions usually possible. Neither of the salt processes seems to offer a fundamental solution of the "complex sulphide" problem.

5. Installations of the volatilization method for oxidized ores and of the roasting and leaching method for one sulphide ore are contemplated. The exact data on the ores and processes in these plants will probably be available at some future date.

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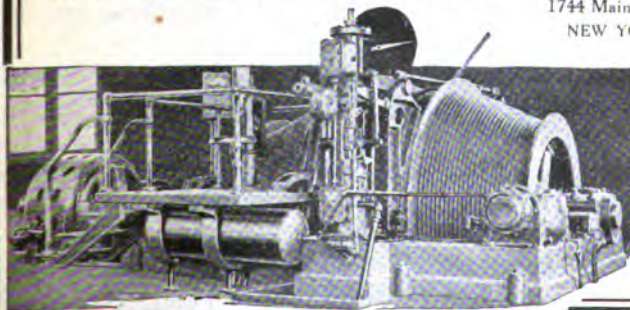
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A great section of exhibits showing the industrial opportunities our South presents in its raw materials will be known as the "Southern Opportunity Section." A "Paper and Pulp Industry Section" has been provided and many elaborate exhibits are in preparation for the paper men when the Technical Association of Pulp and Paper Industry members visit the exposition as they have promised to do again this year. Other exhibits will be of interest to men from the Rubber and Textile Industries. Many more dyestuffs companies have engaged to exhibit their products than formerly. Many of the chemical and allied industry companies have so expanded their operations in the past year, and their products and interests have become so numerous, that they require much additional space to make adequate showings.

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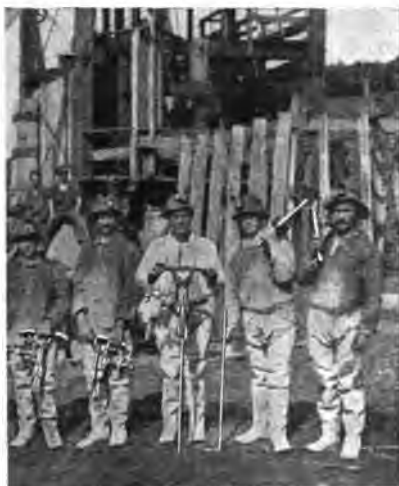
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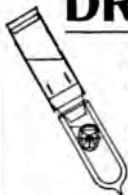
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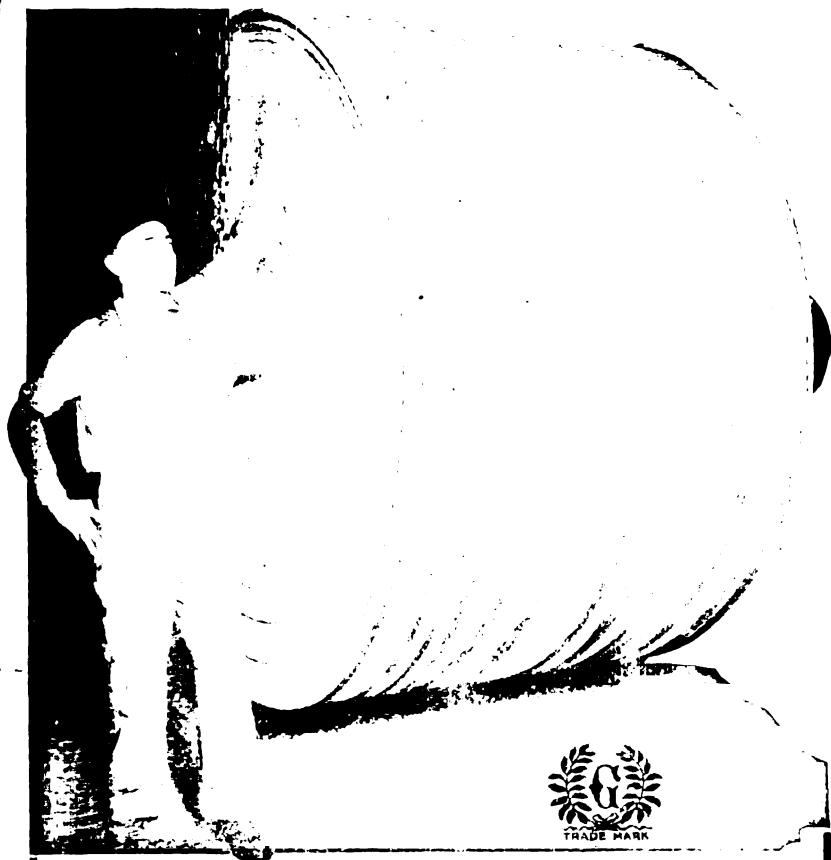
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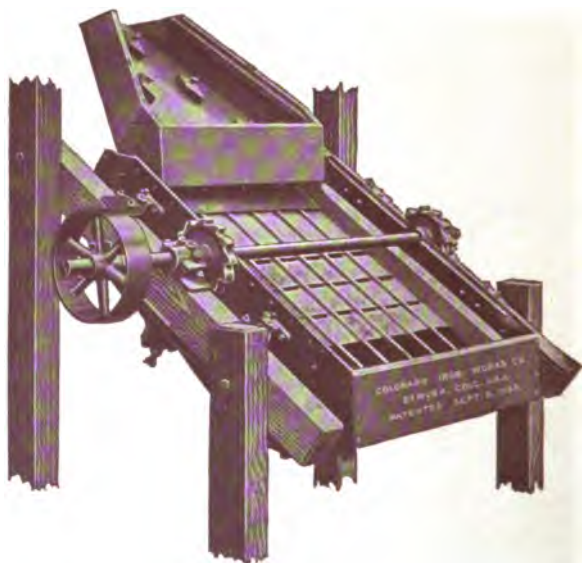
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No. 129.

SEPTEMBER

1917

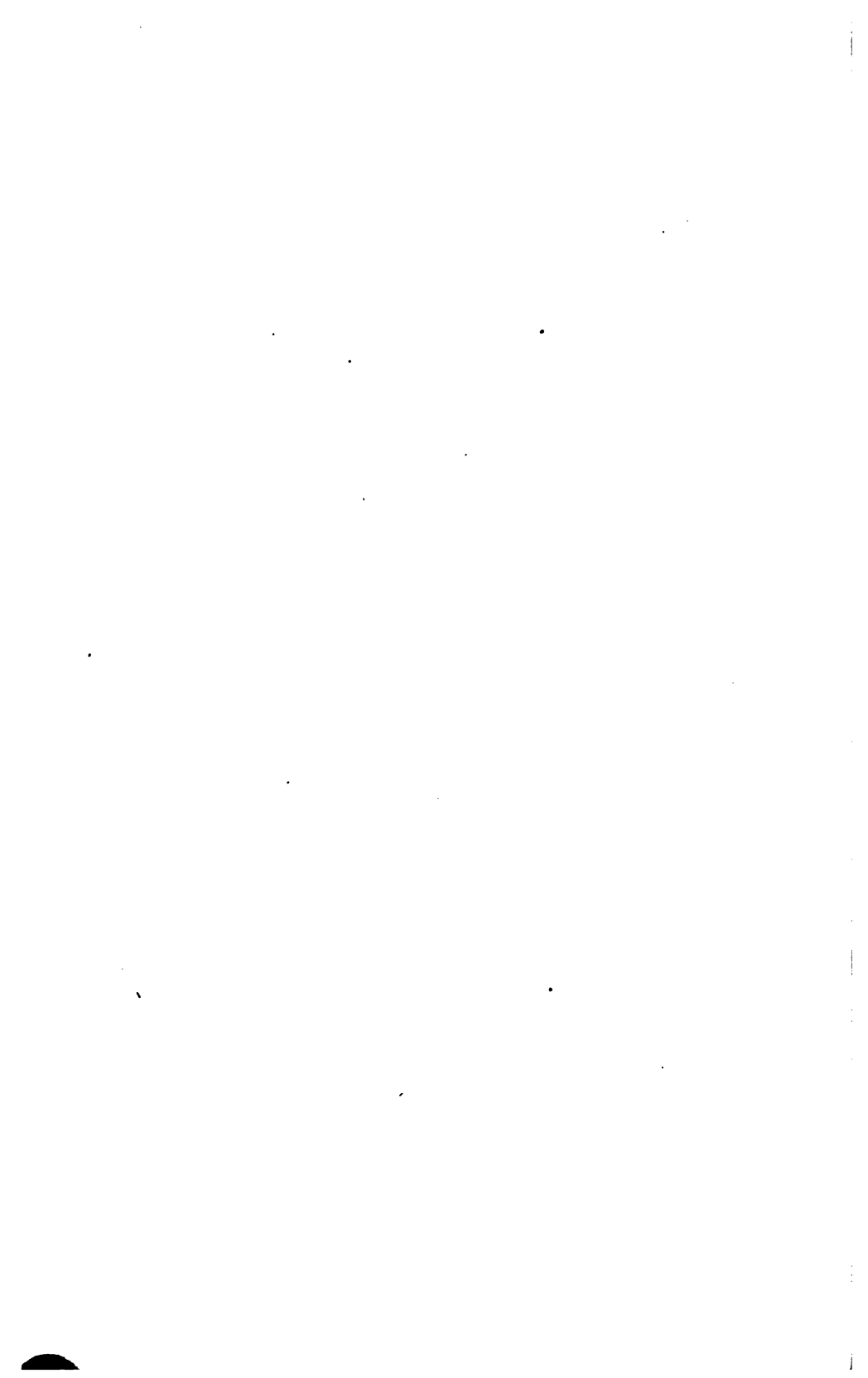
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## NEW YORK SECTION WAR MEETING ON MANGANESE

The first of a series of special war meetings of the New York Section of the American Institute of Mining Engineers was held on the evening of July 20 at the Machinery Club, preceded by an informal dinner. The subject for discussion was the important general one of the manganese supplies of the steel industry of the United States. It had been intended to include chromium supplies but this matter was crowded out and left for a later meeting.

The principal speaker of the evening was Donnel F. Hewett of the U. S. Geological Survey, who, as a result of his recent studies of the resources of the country, ably presented the question of the extent to which the United States can furnish manganese ore.

### ADDRESS OF D. F. HEWETT

The following is a summary of Mr. Hewett's address:

During 1916, when approximately 43,800,000 tons of steel of all grades were produced, 221,532 tons of ferromanganese and 194,002 tons of spiegeleisen were manufactured and 90,928 tons of ferromanganese were imported. If we make the reasonable assumption that stocks at the beginning of the year and the end of the year were approximately the same, 312,000 tons of ferromanganese and 194,000 tons of spiegeleisen were consumed to make this quantity of steel.

The Sub-committee on Ferro-alloys of the Advisory Commission of the Council of National Defense estimates that for the year 1917 more than 45,000,000 tons of steel will be produced and wishes to count on

400,000 tons of ferromanganese and approximately 200,000 tons of spiegel-eisen. In order to make this quantity of ferromanganese, nearly 850,000 tons of high-grade manganese ore with more than 45 per cent. manganese and 500,000 tons of manganiferous iron ore and zinc residuum will be needed. During 1916, the 90,928 tons of ferromanganese, probably made from 200,000 tons of ore, were imported. The imports of ferromanganese in 1917 will be lower than in 1916. Probably, in order to obtain the 400,000 tons of ferromanganese, 700,000 tons of ore must be produced or imported. What part of this necessary ore is likely to be contributed by the United States?

For five years prior to Jan. 1, 1915, the price of 49 per cent. manganese ore ranged from 24 to 26 c. per unit, or approximately \$12.50 a ton, Pittsburgh. Under these conditions, for the same period, the annual production of manganese ore ranged from the minimum of 1664 tons in 1912 to the maximum of 4048 tons in 1913. For the same period, the imports of ore ranged from 176,852 tons in 1911 to 345,090 in 1913. In addition to this ore, large quantities of ferromanganese were imported. For the same period the market price of ferromanganese ranged from \$40 to \$65 a ton. Since war was declared, the price for 49 per cent. ore, delivered in Pittsburgh, has risen steadily to \$1 per unit and the price for ferromanganese has finally attained \$425 to \$450 a ton.

The effect of these increases in price on the number of operators, number of shippers, and production of manganese ore was as follows: For 1912, 8 of the 60 operators reporting to the Survey shipped ore; for 1913, 4 out of 55, and for 1914, 10 out of 75. For 1915, the number of operators reporting was 121, and 31 shipped a total of 9701 tons. The following tables, which were recently prepared in the Geological Survey at the request of the Sub-committee on Ferro-alloys, show the number of operators, number of shippers, and shipments by States of the various grades of manganese ore for 1916 and the first six months of 1917. In connection with the table showing the shipments for the first six months of 1917, it should be noted that inquiries were not sent to several small operators, so that the reduction in number of shippers can not be used to infer the activity of the industry at the present time.

You will notice that the number of operators has steadily increased and the number of shippers for each successive year from 1914 to 1916 has been about three times that of the preceding year. Curiously, the shipments of ore with more than 40 per cent. manganese for 1915, were also about three times those of 1914, the shipments for 1916 about three times those for 1915, and while the shipments for the first six months of 1917 are only slightly in excess of those for 1916, there is a good prospect that the shipments for 1917 will again be three times those for 1916.

It is rather significant that ores ranging from 15 to 40 per cent. manganese, for which there was very little demand as recently as 1912, have been produced in steadily increasing quantities. Thus, the shipments for 1916 were 169,799 tons and for the first six months of 1917, 177,573 tons, with a prospect of nearly 500,000 tons for the entire year 1917. In addition, the production of ores ranging from 5 to 15 per cent. manganese has steadily increased.

From the response shown by domestic shipments to large increases in price, it seems certain that the known deposits in the country will not yield more than a small part, at most possibly 25 or 30 per cent. of the

ores with more than 40 per cent. manganese for the next few years, even under war conditions. It is highly significant, however, that the production of low-grade manganiferous ores, as well as the production and presumably consumption of spiegeleisen, which is made from them, have risen more rapidly than the combined production and imports of ferromanganese, which must be made from high-grade ore. There are other reasons, as well as these, for believing that the steel makers are modifying their practice so as not to require as high a proportion of ferromanganese as formerly.

In order to understand the difficulties in producing the large quantities of high-grade manganese ore that the steel industry now demands, it will serve a purpose at this point to mention briefly the geological conditions surrounding several types of domestic deposits. The important sources of high-grade manganese ore in the world may be considered as belonging to two types, (1) those deposits of rather pure manganese oxide which were laid down between sediments of common types; (2) those deposits of manganese oxide which have been produced by the surface weathering of localized masses of rather pure manganese carbonate or silicate. Examples of the first type are those of the Caucasus Mountains of Russia, which are reported to contain about 80,000,000 tons, and those of the Carrizal district in Chile, which are much smaller. Examples of the second type are those of the Morro da Mina, Brazil, estimated to contain 10,000,000 tons, and several others in the Vizagapatam district, India. It is a coincidence that the United States appears to possess no deposits of the first type and very few of the second type, although there are a number of deposits of manganiferous iron ore in the United States formed by the surface oxidation of localized masses of manganiferous iron carbonate. The manganese oxide of most of the deposits of manganese ore in the United States appears to have been formed by concentration from disseminated carbonate or silicate.

The largest deposit of high-grade manganese ore in the United States appears to be that at Crimora, Va., which in 35 years has yielded about 350,000 tons of ore and may yield a considerable additional quantity. For present purposes, the nodules that make up the ore may be considered as representing a concentration under unique structural conditions of manganese oxide that was originally widely disseminated through the adjacent sediments. It may be considered as representing one of four or five types of favorable structural conditions in deposits that are widespread from Maryland to Alabama. No other deposit in this region has yielded more than 50,000 tons of manganese ore.

Most of the many that are known have been operated and the prospect that they will average more than 10,000 or 15,000 tons each in total production is very small. Most of the deposits of the Batesville district, Arkansas, are somewhat similar to those of Virginia and Georgia, but some differ in details. The size of the deposits is not well known but from the records of production of some of the mines, it appears that they have the same magnitude as those of Virginia.

There are many deposits of high-grade manganese ore in California which appear to represent surface concentrations of disseminated manganese oxide along the outcrops of belts of jasper and chert. So far as available records of production are concerned, it appears that the total yield from a single deposit has not yet exceeded 25,000 tons and some are

TABLE 1.—*Shipments of Manganese and Manganiferous Ore in 1916*

| State              | No. of<br>Inquiries | No. of<br>Replies | Ore Containing 40 Per Cent.<br>Mn or More |               | Ore Containing 15 to 40<br>Per Cent. Mn |               | Ore Containing 5 to 15<br>Per Cent. Mn |               |
|--------------------|---------------------|-------------------|---|---------------|---|---------------|--|---------------|
|                    |                     |                   | No. of<br>Shippers                        | Gross<br>Tons | No. of<br>Shippers                      | Gross<br>Tons | No. of<br>Shippers                     | Gross<br>Tons |
| Alabama.....       | 3                   | 2                 | 1   | (a)           |   |               |  |               |
| Arizona.....       | 6                   | 5                 | 3   | 2,392         | 1                                       | 7,392         |  |               |
| Arkansas.....      | 19                  | 19                | 10  | 6,131         | 7                                       | 3,869         |  |               |
| California.....    | 24                  | 17                | 5   | 6,054         | 2                                       | 221           |  |               |
| Colorado.....      | 8                   | 6                 | 3   | 110           | (b) 1                                   | 90,850        | (d)                                    |               |
| Georgia.....       | 6                   | 6                 | 1   | (a)           | 2                                       | (a)           |  |               |
| Maine.....         | 1                   | 1                 | ...                                       | .....         | 0                                       | 0             |  |               |
| Maryland.....      | 1                   | 1                 | 0   | .....         | 0                                       | 0             |  |               |
| Michigan.....      | 1                   | 1                 | ...                                       | .....         | 2                                       | 37,710        | 6                                      | 202,693       |
| Minnesota.....     | 9                   | 9                 | ...                                       | .....         | 2                                       | (a)           |  |               |
| Montana.....       | 8                   | 8                 | 5   | 3,268         | ...                                     | .....         | (d)                                    |               |
| Nevada.....        | 2                   | 2                 | 1   | (a)           | ...                                     | .....         | 1                                      | 198,817       |
| New Jersey.....    | 1                   | 1                 | ...                                       | .....         | ...                                     | .....         |  |               |
| New Mexico.....    | 4                   | 4                 | ...                                       | .....         | 3                                       | 16,574        |  |               |
| Oklahoma.....      | 1                   | 1                 | ...                                       | .....         | 0                                       | 0             |  |               |
| Pennsylvania.....  | 1                   | 1                 | ...                                       | .....         | ...                                     | .....         | 1                                      | 451           |
| S. Carolina.....   | 1                   | 1                 | 0   | 0             | ...                                     | .....         |  |               |
| Tennessee.....     | 8                   | 8                 | 3   | 212           | 2                                       | (a)           |  |               |
| Texas.....         | 1                   | 1                 | 1   | (a)           | ...                                     | .....         |  |               |
| Utah.....          | 10                  | 10                | 6   | 1,282         | 2                                       | (a)           |  |               |
| Vermont.....       | 1                   | 1                 | ...                                       | .....         | 0                                       | 0             |  |               |
| Virginia.....      | 43                  | 41                | 13  | 4,388         | 7                                       | 5,242         | 2                                      | (a)           |
| Washington.....    | 9                   | 9                 | 1   | (a)           | ...                                     | .....         | 1                                      | (a)           |
| Wisconsin.....     | 1                   | 1                 | ...                                       | .....         | ...                                     | .....         | 1                                      | 3,982         |
| Miscellaneous..... | ...                 | ...               | ...                                       | 2,798         | ...                                     | 7,941         | .....                                  | 30,880        |
| Total.....         | 169                 | 156               | 53  | 26,635        | (b) 31                                  | 169,799       | (d) 12                                 | (d) 436,823   |

(a) Included under "Miscellaneous" in order to keep individual figures confidential.

(b) Not including number of shippers of ore from Leadville to Pueblo for manufacture of spiegeleisen, nor number of shippers of Leadville ore used for fluxing.

(c) Not including Leadville ore used for fluxing.

(d) Ore used for fluxing not included in totals.

TABLE 2.—Shipments of Manganese Ore Jan. 1 to July 1, 1917, with United States Geological Survey Estimate for Entire Year 1917

| State       | No. of Inquiries | No. of Replies | Ore with 40 Per Cent. Mn or More |            |   | Ore with 15 to 40 Per Cent. Mn |             |   | Ore with 5 to 15 Per Cent. Mn |                 |   |
|-------------|------------------|----------------|----------------------------------|------------|---|--------------------------------|-------------|---|-------------------------------|-----------------|---|
|             |                  |                | Jan. 1 to July 1                 |            | U. S. G. S. Estimate for Year, Gross Tons | Jan. 1 to July 1               |             | U. S. G. S. Estimate for Year, Gross Tons | Jan. 1 to July 1              | No. of Shippers | U. S. G. S. Estimate for Year, Gross Tons |
|             |                  |                | No. of Shippers                  | Gross Tons |   | No. of Shippers                | Gross Tons  |   |                               |                 |   |
| Alabama     | 1                | 1              | 1                                | 40         | 200                                       | 1                              | 5,400       | 12,000                                    | .                             | .               | .   |
| Arizona     | 2                | 1              | 1                                | 1,735      | 4,700                                     | 1                              | 3,994       | 7,000                                     | .                             | .               | .   |
| Arkansas    | 16               | 9              | 6                                | 4,361      | 9,500                                     | 4                              | 130         | 430                                       | .                             | .               | .   |
| California  | 22               | 18             | 5                                | 3,012      | 7,300                                     | 2                              | (a) 57,109  | (b) 120,000                               | (b)                           | (b)             | (b)                                       |
| Colorado    | 4                | 2              | 1                                | 100        | 200                                       | 1                              | 32          | 200                                       | .                             | .               | .   |
| Georgia     | 8                | 6              | 1                                | 24         | 300                                       | 1                              | 19,000      | 47,000                                    | .                             | .               | .   |
| Michigan    | 1                | 1              | ..                               | ..         | ..  | 1                              | 76,131      | 250,500                                   | 4                             | 142,710         | 280,000                                   |
| Minnesota   | 10               | 10             | ..                               | ..         | ..  | 3                              | 1,053       | 780                                       | (b)                           | (b)             | (b)                                       |
| Montana     | 7                | 6              | 5                                | 14,206     | 38,500                                    | 1                              | 60          | 150                                       | .                             | .               | .   |
| Nevada      | 2                | 2              | ..                               | ..         | ..  | 1                              | 12,000      | 24,000                                    | .                             | .               | .   |
| New Mexico  | 5                | 5              | ..                               | ..         | ..  | 1                              | ..          | ..  | .                             | .               | .   |
| Oklahoma    | 1                | 1              | ..                               | ..         | ..  | ..                             | ..          | ..  | .                             | .               | .   |
| Oregon      | 2                | 0              | ..                               | ..         | 0   | ..                             | ..          | ..  | .                             | .               | .   |
| S. Carolina | 1                | 1              | 0                                | 0          | 0   | ..                             | ..          | ..  | .                             | .               | .   |
| Tennessee   | 9                | 7              | 2                                | 472        | 1,550                                     | 1                              | 34          | 50  | .                             | .               | .   |
| Texas       | 1                | 1              | 0                                | 0          | 0   | ..                             | ..          | ..  | .                             | .               | .   |
| Utah        | 10               | 7              | 2                                | 690        | 1,625                                     | 1                              | 100         | 200                                       | .                             | .               | .   |
| Virginia    | 32               | 22             | 8                                | 3,705      | 15,430                                    | 3                              | 2,530       | 12,000                                    | .                             | .               | .   |
| Washington  | 9                | 4              | 0                                | 0          | 300                                       | 0                              | 0           | 15  | 1                             | 65,000          | 185,000                                   |
| Wisconsin   | 1                | 1              | ..                               | ..         | ..  | ..                             | ..          | ..  | .                             | .               | .   |
| Total       | 144              | 105            | 32                               | 28,345     | 79,605                                    | (c) 22                         | (b) 177,573 | (b) 474,325                               | (b) 5                         | (b) 207,710     | (b) 445,000                               |

(a) Not ascertained.

(b) Ore used for fluxing not included in totals. Zinc residuum from New Jersey also not included.

(c) Not including number of shippers of ore from Leadville, Colo., to Pueblo, for manufacture of spiegel Eisen nor number of shippers of Leadville ore used for fluxing.

known to have become exhausted after yielding from 5000 to 10,000 tons.

A number of States between the Rocky Mountains and the Pacific Coast have yielded high-grade manganese ore from time to time. The largest production has come from such districts as Tombstone, Ariz., Philipsburg, Mont., and several other districts in New Mexico, Utah, and Nevada, where the manganese oxides are associated with other metalliferous ores. The quantity of ore that may be recovered from the Philipsburg district can not be stated and although reports indicate that the deposits are large and possibly may yield 300,000 to 500,000 tons, most of the known western deposits appear to be small and few appear capable of yielding more than 100,000 tons.

The deposits of manganiferous iron ore are more widespread in the United States and the individual deposits appear to be much larger than those of manganese ore. The greatest production of manganiferous iron ore from a single district (Leadville, Colo.) has been 3,000,000 tons, which includes approximately 500,000 tons smelted to speigeleisen and 2,500,000 tons shipped to lead smelters to be used for flux. In 1915, 52 mines yielded 30,921 tons of ore. During recent years the average composition of yearly shipments from the Leadville district shows manganese to range from 13 to 17 per cent., iron from 33 to 43 per cent., and insoluble from 11 to 12 per cent. There seems to be a prospect that this district will continue to ship from 50,000 to 100,000 tons annually for at least five years, although a close estimate can not be made. Large bodies of manganiferous iron ore have been known for many years to exist in the Pioche district, Nevada, but conditions have permitted their shipment during the last few years only. About 600,000 tons have been shipped recently as flux to lead smelters and the average composition is approximately manganese 14, iron 33, silica 12 per cent. It is reported that the district can yield at least 1,000,000 additional tons of such material.

The sources of the greatest recent production, as well as the largest reserves, of manganiferous iron ore in the United States is the Cuyuna Range, Minnesota. According to E. C. Harder, recent explorations in this district have shown a number of separate deposits, possibly 15, of ore ranging from 1 to 35 per cent. manganese, 20 to 50 per cent. iron, and 8 to 16 per cent. silica. The largest deposit known is estimated to contain about 3,000,000 tons, but the deposits of ore with more than 15 per cent. manganese appear to be smaller and to range from 100,000 to 500,000 tons. It is estimated that the reserves of ore with more than 10 per cent. manganese exceed 4,000,000 tons. Doubtless further exploration by drilling will show the presence of more deposits of similar material, but those now known undoubtedly constitute the greatest localized source of manganiferous iron ore in the United States.

To refer briefly to the manganiferous zinc residuum which is produced from the zinc ores of Franklin Furnace, N. J., it is well known that this material has been the regular source of a large quantity of speigeleisen. During the five years prior to 1915, 553,576 tons of residuum were produced and the annual production has been greater during 1915 and 1916. Although little is known concerning the reserves of ore from which this material is derived, there seems to be good reason for believing that the present rate of production can be maintained for at least five years.



Since this residuum has been melted to spiegeleisen with from 15 to 20 per cent. manganese for a number of years, it is worth while noting at this place that the differences in composition between the New Jersey residuum, the average ore for the Cuyuna district, the average shipments from the Leadville district, and those from Pioche, Nevada, are very slight.

The preceding review aims to outline the position in which the United States finds itself with regard to the supply of manganese-bearing materials during a period when the demand for the production of steel is abnormally high, and there is at least a possibility that foreign supplies of raw material may be cut off. Setting aside the desirability of utilizing domestic resources, to which many of us are partial, it should be apparent under the present difficulty of obtaining ships for ocean transport, that every ton of manganese produced as ore from domestic sources relieves ships to carry at least two tons of materials needed for the actual conduct of war.

A fairly conservative estimate of known domestic sources indicates that if they can be used, they contain enough recoverable manganese to supply the country for about three years at the present rate of production of steel. It is apparent that every effort should be made to utilize our domestic resources.

I shall aim to outline briefly the lines along which efforts may be made. The process of utilization may be considered as made up of four separate processes: Reconnaissance for deposits, mining and milling of ores and metallurgical recovery. At the present time, the Geological Survey plans to maintain four or five geologists in Western States, engaged in the examination of a number of deposits recently reported, as well as others in fields that have been examined previously but have only recently begun to yield a production. Several parties will be maintained in the Eastern States on reconnaissance work and at present it is highly probable that several State Geological Surveys and related organizations in a number of States will carry out similar work.

In this connection the members of the Institute can perform a distinct service by bringing to the attention of the Secretary such information as they have collected in field work, as far as their relations with their clients permit. I regret to contemplate, however, that geologists and mining engineers must be conservative concerning the effect of their many comings and goings upon immediate production, and I feel, therefore, that while this work may greatly aid in permitting an estimate of the actual resources of the country in manganiferous material, it may not greatly effect the rate of production of ores in the next few years.

In the matter of mining and milling of ores, the members of the Institute, as well as the staff of the Bureau of Mines, can perform a great service. On account of the size and grade of several foreign deposits, it appears that only those domestic deposits have been actively exploited during recent years which were capable of being mined in connection with other ores, as at Franklin Furnace, N. J., and Leadville, Colo. As the domestic deposits of high-grade ore have had difficulty in competing with foreign deposits, the business of operating them has yielded little profit and the industry has failed to attract a high class of engineers and workmen. Most mining operations appear to have been conducted with little regard to maximum recovery, the cost of dead work and the necessity for maintaining the means of access.

This conclusion probably applies to most of the districts in Virginia, Tennessee, Georgia and Arkansas. Although at the present rate of production, districts in these States probably supply only several per cent. of the domestic needs of ore, it is not at all impossible that if a high degree of engineering skill were shown in the operations of the mines, the shipments could, within a year, be increased three or even five fold.

I strongly urge the consideration of a plan by which a group of mature engineers, preferably those trained in the Lake Superior region, place their services for a brief period at the disposal of the Committee on War Minerals, recently organized in Washington, in order that they may visit those mines already in operation whose owners show a willingness to accept advice and guidance.

Until about five years ago, the same methods of treating crude ore to produce a high-grade product were in use in Virginia as were used 30 to 40 years ago. The methods were simple and were adapted to the recovery of hard nodules of manganese ore in clay, such as found in most of the deposits. No efforts were made to recover the fragments less than half an inch in diameter. At present close attention to the question of recovery of fine sizes has greatly improved the practice. But the increased recovery from this source will be a small part of our needs.

On the other hand, the prospect is not good for improving by milling processes the grade of material available from manganiferous iron ores. Experiments conducted by Edmund Newton and H. H. Bradt on the Cuyuna ores at the Experiment Station of the Minnesota School of Mines hold out little hope for the recovery of a high-grade manganiferous product from these ores. It is reported also that such work as has been done on the manganiferous iron ores of Pioche, Nev., and Leadville, Colo., similarly does not offer much encouragement. I suspect, therefore, that attention to problems of milling will make but a small contribution to the production of the grade of ores now demanded by the makers of ferro-alloys.

In the matter of the form in which manganese is added to steel, I have recently had the benefit of a long conversation with Mr. Edmund Newton, of the Experiment Station of the Minnesota School of Mines, who has devoted more than a year to the problem of the utilization of the ores of the Cuyuna district. Mr. Newton has considered many phases of the question thoughtfully and thoroughly, and the substance of his conclusions appears to be as follows:

American practice in the manufacture of steel by both Bessemer and open-hearth processes appears to differ from European practice in so far as it relates to the manganese content of the pig iron and the manganese content of the bath before manganese alloys are added.

He has urged, in a recent communication to the Institute, that a great saving in manganese alloys can be achieved if the manganese, in the charge to be smelted to pig iron, is raised to about 3 per cent., so that the pig iron may contain 2 per cent. or more manganese. It is possible to utilize almost all of our domestic manganese and manganiferous iron ores in the manufacture of high-manganese pig iron.

This phase of the question of manganese supply offers a splendid field for constructive experimentation by American steel metallurgists. Much though the industry may need experimental work of this character, I think that the problem will not have been solved until the same metal-

lurgists are able to persuade their more conservative employers that some such methods must be put into practice if the country is to be more independent of foreign supplies of ore, even temporarily. Without doubt, experiments on a working scale in the steel industry are expensive and steel companies will probably show little disposition to change their practice during the period of extraordinary demand for their product. It probably must be left to the metallurgists to see that the utilization of our domestic materials is not delayed until a crisis has appeared. Probably this work offers the most hopeful line of attack in solving the question of domestic manganese supply.

#### ADDRESS OF KIRBY THOMAS

Brazil as a source of manganese ore was fully discussed by Kirby Thomas, mining engineer, New York. He spoke substantially as follows:

The shipment of manganese from Brazil began in 1894, in which year the commercial statistics indicated a shipment of 1430 gross tons. This amount gradually and rapidly increased until in 1900 the shipment amounted to 127,000 tons, and in 1915 to 350,000 tons. The total shipment to and including 1915, amounts to 3,294,871 gross tons.

These shipments went to both American and European consumers. In 1913, the shipments were approximately distributed as follows: United States, 39,400; Great Britain, 16,800; Germany, 5000; Belgium, 11,800. In 1915, United States, 266,877, and Great Britain, 10,000 tons. In 1913, one-fifth of the United States imports came from Brazil; in 1915, nine-tenths.

For some time the principal movement of manganese ores from Brazil was as ballast, and consequently the freight rates were low. To the United States, rates as low as \$3.75 a ton prevailed at times. The average quoted rate for some time was \$5 per ton. Under these conditions, exports of manganese continued to increase.

With the advent of the war, the whole business was disjoined and readjustment followed. Shipments to the United States were increased, Germany's shipments ceased entirely, and in general, the shipments to Europe were curtailed. Shipments of Brazilian manganese to the United States during 1916 were over 500,000 gross tons. In 1917 they will probably be less, owing to the difficulty in securing ships.

The freight rates which have prevailed since the war have been extremely high, \$10 during the early part of the year, increasing to \$15, and commitments on considerable tonnage, at a rate as high as \$20 a ton have been under negotiation within the recent months, due entirely to the war conditions. The high price paid for the transport of coffee and hides to the United States has made the shipment of manganese, except small amounts, unattractive, and the United States has been compelled to furnish its own transportation.

At present the supply of manganese from Brazil is controlled by the factor of ocean transport. There is, however, another factor—the question of railroad transportation to the seaport.

The Brazilian manganese deposits lie in the southern part of the State of Minas Geraes, north from Rio de Janeiro 300 to 400 miles, and are reached by the Brazilian Central Railroad. The main branch of this

line continues through the country beyond the manganese operations. In the mountains it branches off in several directions, furnishing a general feeder system. Over this road all the agricultural products find their way to the important market at Rio. This city of 1,000,000 people necessarily consumes a large amount of agricultural products. The railroad, therefore, is a vital and controlling link in the transporting of bulk material, such as manganese ore. It is owned by the Government, is inefficiently managed, and its construction is hardly up to American standards. The road is broad gage and has considerable freight equipment, rather light. The engines are all of American make, except some Manheimer engines which were put on a few years ago.

Recently a combination of large manganese interests advanced to the Government, which is more or less in financial straits, about \$500,000, which was to be utilized to supply additional equipment to increase the shipment of manganese. Under this arrangement, these manganese producers received special rates and considerations for a limited period. This new equipment, which is now installed, enabled the road to bring down 500,000 tons of ore, a distance of 350 miles to the port of Rio, during the past year. The railroad at present is practically at its limit, although competent traffic men consider that with a little more efficiency and speeding up the present equipment could transport 150,000 to 200,000 tons more annually, making a total of from 600,000 to 700,000 tons. This road, then, is the first controlling factor in the manganese supply for present purposes. This fact has been recognized by producers and consumers, and by the Government. I am advised of this by exchange of communications between Washington and Rio de Janeiro, and as a matter of fact, the Brazilian Government installed an effective system of guards on the railroad during January, and prior to the breaking of relations with Germany.

The next factor affecting the transport of the ore lies in the limited and expensive facilities of loading at the harbor in Rio, which has no facilities for loading bulk ores and the transport of the material from the shore to the ship is under control of a hindering concession. The result is that a very substantial additional charge in the transportation chain is involved here, and more particularly, the question of storage and providing quick facilities for loading and for receiving the bulk ores, are lacking.

The port conditions are such that it is necessary to unload the ore from the cars on to lighters, transport it to islands in the harbor and from there transport it to lighters or ships. A plan has been under way to construct a dock, but this has not materialized, chiefly because of the failure of the leading interest to cooperate and coordinate its activity and partly because it involves a considerable investment. The engineering details of this plan have been worked out by the Pearson Engineering Company and some movement to provide the docks and facilities in connection with the large coal docks installed by the Rio Janeiro Tramway Company has been made.

There are two belts of manganese deposits, which extend for a distance of 100 miles, across the southern part of the State of Minas Geraes. The lower part of the belt is somewhat analogous to the Potomac-Staunton River deposits in Virginia. The deposits are irregular as to character and extent, but are larger than any we have developed in this country,

by far. The northern belt extends from Miguel Burnier west to Ouro Preto and presents prospects of very large tonnage. This also varies in grade and character.

As to the nature of the deposits and actual and prospective tonnage available, this can only be discussed in general. Messrs. Singewald and Miller estimate 10,000,000 tons in the Mina de Mora mine. Another proposition involves 2,000,000 tons and another has an actual tonnage of 500,000 tons. In my opinion, by opening up and developing these Brazilian deposits, an adequate supply of manganese could be provided for the United States for 50 years to come. This, of course, is a generalization. For the immediate consideration of the manganese problem, I think we may assume that a supply could be developed within a few months, which would be sufficient to supply the United States during the period of the war and for several years after.

Some of the more available properties are controlled by local operators, or under contract to representatives of American consumers. There are, however, numerous other properties, equally as good, but only slightly developed, which are available.

The railroad system can be rendered more effective by increased equipment and more efficient operation and I think the Brazilian Government would be interested in coöperating in such a plan. There is an opportunity to extend railroads already built into the field. Such a project would deliver the ore at a harbor west of Rio, which is suitable for development. This plan has been considered, but has not yet been undertaken.

The question of shipping is largely up to the political governments that control the high seas. The individual is quite helpless in the face of the overwhelming domination of political force in the matter of ocean transportation. The American Government is vitally interested in securing transportation for manganese from Brazil to the United States and also in returning to Brazil coal and general products, which are absolutely needed in Brazil; hence the Brazilian Government is also interested in increased facilities for ocean transportation.

England has already made a proposition to purchase coffee from Brazil, on condition that Brazil will transport a certain amount of food products to England. The United States has made a similar proposition, to utilize the boats sailing from Brazil to the United States, and this would go a long way to solving the manganese problem. It is possible also that some of the boats that the American Government is building can be applied to this service. I think that this Government should undertake to supply a certain regular tonnage for manganese and this tonnage should be apportioned to consumers in the American field, particularly those engaged in the essential phases of iron industry, the Government to exact a reasonable ocean freight and to assure shipping facilities during the war. The subject is of such importance that it justifies the consideration and action of the highest political authorities. This has also been recognized, and there has been an exchange of information on the subject.

The general content of the manganese ore is between 45 and 50 per cent. metallic manganese. This product could be converted into ferromanganese, either at Rio or in the field, which would simplify transportation. To this end I have compiled some data.

First, in regard to the question of electric smelting. The power can

be obtained from the Rio Tramway Co. at a fairly reasonable rate and the installation on short notice of an electric smelting unit here, or several units, is entirely practicable. Water power in the interior is available and can easily be developed at several places, so that electric smelting in the interior is also practicable.

These electric smelting units, if not found profitable after the war, can be utilized for producing cast steel, from the large amount of scrap steel available, or possibly for the conversion of high-grade iron ores of the interior into iron and steel. There is practically no production of iron in Brazil and the demand is large and growing. The question of charcoal smelting and blast furnaces, to produce ferro-manganese, has also been considered.

It would seem that this Institute could effectively coöperate with various official departments and committees and committees of defence, by undertaking to supply the technical and physical information to solve the problems above outlined.

I would recommend that at this meeting we consider the question of appointing such a committee, to include a civil engineer, to take the data available on electric smelting; and also a metallurgist, who would consider the question of charcoal smelting of manganese; and members familiar with the general operating conditions in the Latin-American countries and with the transportation between the Latin-American countries and the United States.

Discussing deposits of manganese ore in Costa Rica, Mr. Thomas said that they were very promising on the west coast, but that while shipments already made had not been of a satisfactory grade, he had been assured that good material exists in reasonably extensive quantities in that district. One disadvantage is that the ore has to be loaded on lighters and carried to vessels some distance from the shore. Other deposits in that country are under investigation and will probably turn out to be important. The Panama deposits, he said, are also promising.

#### GENERAL DISCUSSION

The metallurgy of the production of ferromanganese was presented by J. E. Johnson, Jr., consulting engineer, New York. It is an acknowledged fact, he said, that more heat is necessary in the blast furnace to smelt manganese ore than iron ore, as much as two times and possibly three times more. More carbon is necessary, partly because of the fact that carbon has a greater affinity for manganese than manganese has for iron. As a result, in making both spiegeleisen and ferromanganese, the carbon absorbed and existing as a constituent of these products is greater than in the case of making pig iron. In spiegeleisen the percentage of carbon is around 5 per cent., and in ferromanganese around 6 per cent. It is practically impossible to use spiegeleisen as a substitute for ferromanganese in making steels relatively low in carbon, because in adding the spiegeleisen, so great a percentage by weight of carbon is added at the same time. Because of the bulk necessary to obtain the desired amount of manganese, the spiegeleisen must be added hot in most cases, and while this practice is entirely feasible in rail making, present steel plants are not laid out to do this generally. It is an impossibility to meet certain specifications when using spiegeleisen, which demand certain carbon and manganese percentages.

The rôle of manganese is largely as a deoxidizer, the speaker said. Experiments to use ferrosilicon as a substitute have not been very successful, at least to no greater extent than as a substitute for about 25 per cent. of the ferromanganese. The silicon appears to be less efficient as a deoxidizer, and it has been his experience that oxygen remains in metal when 2 or 3 per cent. of silicon is present, whereas it would immediately vanish in the presence of 1 per cent. of manganese. Aluminum as a deoxidizer could not be used in large quantities. Another important consideration is the fact that manganese seems to affect the rolling properties of the steel. Mr. Johnson does not believe that any adequate substitute is possible.

In the metallurgy of ferromanganese the question of silica in the ore is an important one. Manganese is more active than iron. Under certain temperature conditions, the manganese is easily lost, whereas under other conditions of heat other serious results occur. It was stated at one time that it required 120 units of manganese to make 80 per cent. ferromanganese or a loss of 30 per cent., although Mr. Johnson said that he understood that this loss had in some practice been reduced to 6 per cent.

Discussing the use of charcoal as a fuel in making ferromanganese, he said that theoretically it was not desirable, but from his own experience in smelting manganese ores from the Cuyuna range, he had been able to make spiegeleisen with a low fuel consumption, much to his surprise.

Bradley Stoughton, Secretary of the Institute, discussed the effect of manganese on steel. As an explanation of the apparent fact that a substitute cannot be found for manganese, the speaker said that manganese takes oxygen completely out of the steel, whereas silicon often remains in the steel as silica and aluminum as aluminum oxide or alumina. Manganese not only takes the oxygen out, but it also removes the silica as a silicate. Oxide of manganese remaining possibly in the steel is also not as harmful. An important feature, also, is the neutralization of sulphur by manganese.

Much has been said concerning German substitutes for manganese, but Mr. Stoughton does not believe these statements. Reports coming to him lead him to believe that German steel as at present made is low in manganese and poor in quality. If the German situation is bad, said Mr. Stoughton, our situation would be still worse under similar circumstances.

Mr. Stoughton believes that there might be many ways of solving this important question. The possibility of the electrolytic production of pure manganese from manganese dissolved with iron salts had been suggested, the manganese going to the cathode. Possibly pure manganese could be formed from low-grade ores existing in this country in large quantities. He believes the real problem is a method of using these ores. He said that any suggestions that members of the Institute might have, no matter of what nature, if forwarded to him would be brought to the attention of the authorities at Washington and carefully investigated.

Henry D. Hibbard, consulting engineer, Plainfield, N. J., said that many years ago, at a suggestion from P. H. Dudley of the New York Central Railroad, he had attempted to replace manganese in steel by silicon. In 1885, he made a steel of about 0.30 per cent. carbon, containing 0.25 to 0.30 per cent. silicon. The addition of the silicon, of course, completely killed the steel, but the ingots were red short.

Mr. Hewett again brought up the question of the control of manganese in the steel bath, saying that it had been suggested to him that this could be done before the regular additions of ferromanganese and that American practice differed from that of Germany and some other countries. In American steel making, he understood that the manganese in the bath toward the end of the heat was as low as 0.10 or less, whereas in Germany it was maintained as high as 0.40 or 0.50. He asked whether the use of high-manganese pig iron would not tend to maintain a higher percentage of manganese in the bath as suggested in the foregoing as well as prevent a certain amount of oxidation and therefore the necessity of using so much ferromanganese.

Mr. J. E. Johnson, Jr., said that for a long time it has been acknowledged that better steel is made from pig iron containing manganese than from pig iron containing none. He did not see how it was possible in present American practice to maintain the manganese at a high point near the end of the heat. In Bessemer practice the manganese is reduced to 0.10 before the carbon goes, but in open-hearth practice it might be possible to keep up the manganese to some extent.

Henry D. Hibbard said that in Sweden, pig iron, containing 3 to 4 per cent. manganese, had been used in Bessemer practice so that the steel made contained enough manganese without any further addition to make good steel. This fact, he said, had really saved the life of the Bessemer process, which otherwise would have had to be abandoned because in England its use without the presence or the addition of manganese had had disastrous results.

Prof. G. A. Roush of Lehigh University, South Bethlehem, Pa., gave some interesting testimony regarding the sequence of the elimination of various elements in Swedish and American practice in both Bessemer and open-hearth steel making and also brought up the question of the use of a calcium-silicon-aluminum alloy. It had been advocated that this compound, when acting as a deoxidizer, at the same time formed a slag which removed itself easily from the steel. What is technically known as "Swedish operation" in the Bessemer converter, descriptions of which can be found in such works as Harbord's *Metallurgy of Steel*, makes use of manganese in the raw metal for self-killing the charge. If our present metallurgical processes in the open-hearth furnace can be modified to work along the same line, it will mean an enormous saving of high-grade manganese resources.

Some facts as to the actual supply of ferromanganese for the steel industry of the United States at the present time were given by Edwin F. Cone, associate editor *The Iron Age*. It is a fair assumption, he said, that the output of steel in this country in 1917 will not exceed 45,000,000 tons. Assuming the average amount of ferromanganese necessary in making various grades of steel to be 17 lb. to the ton, and taking into account the fact that probably two-fifths of the Bessemer steel output can be satisfied entirely by the use of spiegeleisen, the amount of ferromanganese necessary for the steel industry of the United States in 1917 would be about 316,000 gross tons, of which 10,000 tons is used by the iron foundries of the country.

The alloy committee of the Council of National Defense in a recent canvas of the situation, has determined that the steel industry at present consumes 28,000 tons of ferromanganese per month. He said the ques-



tion of the British supply had not been brought into the discussion but it is an important one. So far this year the domestic production of ferromanganese, according to the blast-furnace returns of *The Iron Age*, has averaged 20,000 tons per month. This was increased to 21,000 tons in June, but it is hardly probable that this figure will be greatly exceeded. Therefore, to meet the calculations of the alloy committee, nearly 8000 tons of alloy must be imported. Statistics so far this year indicate that this supplementary amount is not being supplied. While in 1916 British imports had averaged 7500 tons per month, and while for the first four months in 1917 the imports had been about 6000 tons per month, in the last two months these have been under 3000 tons, with only slightly over 2000 tons in May. The sinking of over 2000 tons by submarines in the last six weeks was mentioned as showing the seriousness of the shipping situation.

In a written discussion of the general topic, Chester D. Tripp, of the Miami Metals Co., Chicago, said in part:

The manganese industry, one of the very vitals of the steel business, is being fostered and stimulated by famine prices into trying to find itself and make itself permanent on our shores. As usual, we have followed in the past the course of least resistance and have bought the foreign goods, paying little attention to what we would do without it or how the other side procured its ore material to supply us with the finished article.

We can safely say that there is no chance of producing in the United States, a natural unprepared and satisfactory product that would ever even approach in price what can be had from India or South America on the wage scales obtaining in those places.

The next question is, what can be done to remedy the situation. Only three alternatives present themselves:

*First.*—Arbitrarily putting the price of ore so high at the time of a national emergency, such as the present, that not only the best deposits, but even second and third class properties can yield a profit to the operators.

*Second.*—Trying to educate the steel trade to use intermediate grades of manganese metal, the idea being to get approximately the same results with high-grade spiegels instead of ferromanganese, thus allowing the introduction of a large quantity of ore which today is considered impossible to utilize.

*Third.*—Working out some systems of benefaction which would improve some of the siliceous orebodies of the West so that they would be fair furnace material.

The first scheme brands itself as unworkable. We have had two years of excessive prices and still production is not what it ought to be. My opinion is that it would take months of time to create a real domestic production even though the prices of ore were twice as high as they are today.

The second alternative has real possibilities. There is no reason why large quantities of intermediate grades of ferro-alloy could not be disposed of without bad effects. Our requirements are largely the result of our education, and our education is the result of the goods we are able to most easily procure. The imported material has almost always been based on two grades—20 per cent. spiegel and 80 per cent. ferro. There has been very little offered of the available intermediate grade, and naturally very few people look on it with favor.

This is not the case in Europe. On the other side all grades of manganese are produced and used. There are many departments in the steel business that would actually be very much better off using 40 or 60 per cent. alloy than they would be with 80 per cent., particularly where the phosphorus is low, as in the average American ore. There is a real possibility for educational work along this line that has been seriously neglected, but which should be studied and generally understood. For it is possible that the next time ferromanganese becomes scarce in this country, it will be the result of a complete blockade of our coasts, which would practically paralyze our steel business, as things stand today.

The third alternative, "benefaction," is receiving considerable attention in this country now. There are few localities that could hope over a period of years to concentrate manganese ore and do it commercially, particularly if they have to make a product that will make 80 per cent. material, but there are localities in the country that could, with no great expense, produce thousands of tons of reasonably good grades of manganese ore on a high market.

As to concentration, it is practically impossible to separate manganese from iron by the jiggling or concentrating process. Such work is almost a laboratory problem. It can be done, but it is so expensive as far as first cost goes, that it probably will not be attempted until the market permanently reaches a higher basis. My feeling is that the great hope in the matter of concentration is the removal of silica, so that the slag volume on a blast furnace can be kept within reasonable limits.

There are three important general types that lend themselves to this treatment:

*First.*—The manganese-bearing clays and schists of the Appalachian Range.

*Second.*—The ferruginous iron ores of the Lake Superior country.

*Third.*—The manganese gossans of the Rocky Mountain country.

Much work has been done on the first of these groups, with more or less satisfactory results.

In the second group, namely, the Lake Superior, no real work of concentration has been done, but the territory contains enormous deposits of highly manganeseiferous iron ore which, except for silica, will be of great value in famine times.

The third group, the Rocky Mountain territory, is beginning to get its share of attention, but no important developments have yet taken place. I have every reason to believe that considerable quantities of a fair grade of manganese ore could be worked out by concentration at deposits which are almost valueless today, and much of the work could be done with equipment already in existence.

No greater encouragement could be given to American ore producers than by the development in the steel industry of a fairly steady market for intermediate grades of alloy. For instance, it avails a man very little today to work out an ore that will make a 50 or 60 per cent. metal, unless he can be paid for his efforts during the term of the war. He is face to face with the fact, as things stand now, that a little later he will have a hard time selling his product at a living price.

Marcel Daly submitted the following in a written discussion:

I do not think that North America, including the United States and Canada, will ever be a favorable ground for manganese mining and man-

ganese production. It does not look like a petrological province for this kind of ore. As far as I know, with the exception of a few scattered deposits, manganese ores in the United States, on account of their character, require expensive mining and concentration. The result is that although the present high prices of manganese would allow the working of some low-grade ores, private capital is not likely to enter this line of business on any larger scale than it does now, as the future is most uncertain. As soon as the war is ended and importations are resumed, prices are liable to drop to a point where home production will almost cease to be profitable. For manganese, in my opinion, the United States will have always to rely essentially on importation.

It seems to me that this war has created exceptional conditions which would have to be handled by exceptional means, and I personally would like to see the Government take the matter in hand. It would be justified in taking up and working manganese deposits under conditions that would not allow private interests to do so. This war is waged with steel. A deficiency in steel means an increase in the loss of lives, an increase in pensions, etc.

Our principal contributor now is Brazil, where there seems to exist a field of practically unlimited extent. Why not concentrate our efforts there? Transportation of the ores from Brazil to the United States ought to be assured by the Government itself or at least done under its control. There are some German interned ships in Brazil (about 150,000 tons) which may help in the matter. I do not see why Brazil would not be able to furnish the United States with all the necessary manganese during this war, or at least the greater part of it.

### **AID ASKED ON THE SUBMARINE PROBLEM**

The Naval Consulting Board is asking the aid of all the members of the Institute in its effort to help solve the submarine problem. Probably every member of the Institute has some suggestion, and while each one of these may be of only slight value in furnishing the remedy, the aggregate may be of great service. Every member of the Institute will soon receive a pamphlet known as Bulletin No. 1 of the Naval Consulting Board of the United States in which full information and instructions will be given to all of those who are willing to assist. It is hoped when this information is received the members of the Institute will give it their prompt attention and write as soon as possible to the Secretary of the Naval Consulting Board, 13 Park Row, New York, N. Y.

### **MEETING OF THE EXECUTIVE COMMITTEE OF THE BOARD OF DIRECTORS, JULY, 1917**

The following persons were appointed as representatives of this Institute on the American Committee on Engineering Standards: George C. Stone, J. W. Richards, Arthur L. Walker.

The Secretary presented a request from the Engineering Council that the representatives of the Institute be arranged in the order of retirement so that one would retire in 1918, two in 1919, and two in 1920. The

Committee decided that the preference should be made by drawing lots and the lots resulted as follows:

Retiring in 1918, J. Parke Channing;

Retiring in 1919, S. J. Jennings and B. B. Lawrence;

Retiring in 1920, Edwin Ludlow and P. N. Moore.

J. E. Johnson, Jr., was appointed Chairman of the Committee on Arrangements of the New York Meeting, February, 1918.

Sixty-eight Members, six Associates, and eighteen Junior Members were elected. Two members were reinstated.

RESOLVED, that the dues of members on active service in the war shall be remitted when and if a desire is expressed by the member or by his representative.

The sum of \$150 was appropriated to the New York Section.

The contract for soliciting advertising was authorized with Mr. E. G. Nellis.

### THE WOMAN'S AUXILIARY TO THE A. I. M. E.

MRS. SIDNEY J. JENNINGS, *President*

MRS. ARTHUR S. DWIGHT, *First Vice-President*

MRS. KARL EILERS, *Second Vice-President*

MRS. H. W. HARDINGE, *Third Vice-President*

MRS. BRADLEY STOUGHTON, *Recording Secretary*

MRS. AXEL O. IHLSENG, *Corresponding Secretary*

MRS. HARRIS K. MASTERS, *Treasurer*

#### *Welfare Committee*

MRS. C. C. BURGER, *Chairman*

#### *Sub-Committees*

##### *Americanization*

MRS. LEVI HOLBROOK, *Chairman*

##### *Belgian Relief*

MRS. HENRY H. KNOX, *Chairman*

##### *Emergency*

[MRS. HERMAN A. PROSSER, *Chairman*

A very successful meeting of the Council of the Woman's Auxiliary was held Wednesday, July 18, and plans for war relief work were thoroughly discussed.

The Belgian Relief Committee has sent out the following letter to the ten local sub-committees, which were formed to raise money to feed the Belgian children:

"The Commission for Relief in Belgium announced in a letter from Mr. Hoover that after the first of June the United States Government would take over the relief of Belgium and Northern France and called attention to the following paragraph:

'Naturally, having built up such an effective organization you will desire to keep it alive as far as possible, and we venture to suggest that, although the general relief of the countries involved will now be met by the Government appropriations, emergencies and special conditions may arise which could only be met by private donations. In such circumstances your organization will afford a ready means of meeting the demands of the situation, whatever they may be. Should any of your contributors

desire to continue their gifts, notwithstanding the present position, they may be assured that their contributions will be expended sooner or later to great advantage, since in any event relief in many forms will doubtless be required after the war.

"From this you will see if you care to continue your contributions for Belgian Relief, the Commission will accept them. Also, your attention is called to the Belgian Kiddies stock which is still being subscribed to and which is peculiarly the contribution of the Mining Engineers.

"This Committee has in no wise abandoned the work of Belgian Relief but for the present has been authorized to take up some other form of Foreign War Relief with the understanding that when the Commission for Relief in Belgium needs its exclusive services again it has only to call upon it.

"In a multitude of objects each one making a strong appeal to our sympathies it is difficult to determine for which one we will work, but as the U. S. Army Engineers are about to undertake the task of reconstruction in Northern France, to aid in that work might seem the best field for our activities. In every case where we are supplying a man or woman with means of self support we are giving constructive help.

"We propose raising money for the following objects:

I. To purchase agricultural implements and artisans' tools, for it must be remembered that in the devastated districts all tools have been destroyed, and the French Government earnestly desires that these shall be supplied.

II. Money for the relief of the destitute.

III. Clothing for women and children, to be made after patterns which can be had on application to this Committee.

"Will you kindly let us know if your Committee will continue the work it has so efficiently done for any of these objects?

"This Committee extends its hearty thanks and appreciation to all those who have so splendidly responded to its appeal for Belgian children and trusts that it may have the same generous support in its new work.

Yours faithfully,

Sophie C. Knox (Mrs. Henry H.)  
Chairman."

The Acting Chairman of the Emergency Committee, Mrs. S. G. Weinberg, submitted the following report:

"The Emergency Committee was formed for the purpose of being ready to take up any work that might come up through sudden calamities or unexpected emergencies and where, more especially, the wives of mining engineers might be in a position to help. At present, the particular aim of the Committee is toward help for our own Engineer Regiments in France in organizing recreation and rest stations and furthering any other need that may arise. A member of the Committee is about to sail for France to report upon what is more particularly needed."

Since then, Mrs. Arthur S. Dwight, the member referred to, has sailed for France, leaving on Tuesday, Aug. 7, on the S. S. "Touraine." She took with her

- 300 lb. of candy,
- 150 quarts of fruit jelly,
- 400 lbs. of assorted groceries,
- 1200 bags of Durham tobacco,
- \$100 worth of surgical plaster,
- 50 "comfort bags," each containing the following articles:
  - 1 comb, 1 tooth brush, 1 tube tooth paste, 1 cake soap, 1 wash cloth, 1 towel, 1 handkerchief, 1 small mirror, 1 blunt scissors,
  - 2 prs. shoe-laces, coarse black and white sewing thread, darning cotton, needles, buttons, thimble, pins, plain and safety, paper and envelopes, pencil.

These bags cost \$2 each to equip and were the gift of the Century Theatre Club, of New York, of which Mrs. Axel O. Ihlseng is president. Mrs. Dwight will use them among the Engineers at her discretion. The club will have more ready to send as soon as Mrs. Dwight requires them. If any member of the Auxiliary wishes to help in this special line of work, i.e., furnishing comfort bags, please communicate with Mrs. Ihlseng.

Mrs. Dwight has a personal letter to the head of the Red Cross in France, and with his assistance will look over the ground where the Engineers expect to be located, and if feasible will plan some special relief work for the Woman's Auxiliary. This, however, need not interfere with any relief work now under way.

Mrs. Harris K. Masters was unanimously elected treasurer of the Auxiliary to take the place of Mrs. Dwight, who was acting treasurer.

All ladies in the families of members of the A. I. M. E. are cordially requested to join the Auxiliary.

SUSANNE M. IHLSENG (Mrs. Axel O.)

*Corresponding Secretary.*

## MEMBERS OF THE INSTITUTE IN MILITARY SERVICE

- AARONS, J. BOYD, Major.  
 AGNEW, FRANK VANS.  
 ALLEN, HERMAN H., Private, D Co., 8th Reserve Engineers.  
 BAINS, THOMAS M., Major, Engineer Officers' Reserve Corps.  
 BANKS, H. R., Lieutenant, Canadian Engineers, B. E. F.  
 BARBOUR, PERCY E., Deputy Supt. (Captain), New York State Troopers; Captain, 22d Regiment, N. Y. N. G.  
 BARLING, H. B., 1st Lieutenant, Engineer Officers' Reserve Corps.  
 BIGELOW, BRAXTON.  
 BLOOMFIELD, E. C., Lieutenant.  
 BORROW, F. K.  
 BROCK, R. W., Major, 19th Reserve Battalion of Canadians.  
 BROOKS, ALFRED H., Captain, Engineer Officers' Reserve Corps.  
 BROWNE, DE COURCY, 1st Lieutenant, Ordnance Dept., Fortress Monroe, Va.  
 BULLOCK, S. C., Captain, Royal Engineers, British Army.  
 CADMAN, JOHN, Liason Officer, B. E. F.  
 CAETANI, GELASIO, 1st Lieutenant, Royal Italian Engineers.  
 CAHEN, JAMES P., JR., Reserve Officer, U. S. Cavalry.  
 CALDWELL, WILLIAM A.  
 CHAMBOST, P. J. E. E.  
 CHAPMAN, R. H., Major, Engineer Officers' Reserve Corps.  
 CHURCH, JOHN L., Captain, Engineer Officers' Reserve Corps.  
 CRAWFORD, T. R., Engineer Officers' Reserve Corps, Ft. Riley, Kans.  
 DAVIS, A. W.  
 DOUGALL, RALPH. Killed in action.  
 DUDLEY, BOYD, JR., Captain, Ordnance Dept., Engineer Officers' Reserve Corps.  
 DUNN, THEODORE S., 1st Lieutenant, Engineer Officers' Reserve Corps.  
 DWIGHT, ARTHUR S., Major, Engineer Officers' Reserve Corps.  
 EDWARDS-LECKIE, R. G., General.

- EMMERSON, ERIC ROY.  
EVANS, A. W., Lieutenant Colonel.  
FAIRBAIRN, GEORGE, Captain.  
FARRANT, JAMES C.  
FIDLAR, J. B., 2d Battery, Officers' Training Camp, Fort Logan, Ark.  
FORBES, CARROLL R., Captain, Engineer Officers' Reserve Corps.  
GATES, A. O., Lieutenant, U. S. Naval Reserve.  
GILCHRIST, A. D.  
GOODSPEED, G. E., JR., 1st Lieutenant, Engineer Officers' Reserve Corps.  
GORDON, ERNEST.  
GORMAN, THOMAS C.  
GRAVATT, C. MARSHALL, Aviation Corps, Columbus, Ohio.  
GRUNOW, W. R., 1st Lieutenant, Engineer Officers' Reserve Corps.  
GRUNSKY, CARL E., JR., Captain, Engineer Officers' Reserve Corps.  
HAGUE, WILLIAM, 1st Lieutenant, Engineer Officers' Reserve Corps.  
HALL, WILLIAM T.  
HARDINGE, HARLOWE, 1st Lieutenant, Signal Corps, U. S. R.  
HAYES, F. H., Captain and Adjutant, First Arizona Infantry.  
HEAD, JAMES L., 2d Lieutenant, Engineer Officers' Reserve Corps.  
HEGGM, ALFRED G., Captain, Engineer Officers' Reserve Corps.  
HIBBERT, A., Lieutenant, 3d Canadian Tunnelling Corps, B. E. F.  
HINES, P. R., 1st Lieutenant, Engineer Officers' Reserve Corps.  
HOLLOWAY, W. S.  
HURD, RUKARD, Major, Engineer Officers' Reserve Corps.  
HUTCHISON, WILLIAM, Canadian Forestry Battalion, France.  
IRVING, J. D., Captain, Engineer Officers' Reserve Corps.  
KIRKPATRICK, G. H., Colonel.  
KNAPP, ARTHUR, Captain, Engineer Officers' Reserve Corps.  
KOBBE, WILLIAM H., 1st Lieutenant, Engineer Officers' Reserve Corps.  
KOPÉ, G. S.  
LACROIX, M. F., Captain, Engineer Officers' Reserve Corps.  
LAKELAND, WILLIAM J.  
LAVERY, VAUGHAN M., Lieutenant, 3d Tunnelling Corps, Canadian Engineers, B. E. F.  
LEONARD, W. H., Captain, Troop E, 1st Regiment, Colorado Cavalry.  
LOVE, ROBERT A., 1st Lieutenant, Engineer Officers' Reserve Corps.  
LUNN, R., JR.  
MACNUTT, C. H., Canadian Engineers' Training Depot, Crowborough, Sussex, England.  
MAGEE, JAMES F., 1st Regiment of Engineers, U. S. Expeditionary Force in France.  
MEWHIRTER, SYDNEY A., Designate Lieut., Corps of Engineers, U. S. Reserve.  
MILLARD, WM. J., Captain, 12th Engineers, U. S. R.  
MORGAN, G. H., Lieutenant, Engineer Officers' Reserve Corps.  
MUDD, SEELEY W., Major, Engineer Officers' Reserve Corps.  
NEWBERRY, R. W.  
OSBORNE, CHARLES G., Red Cross Council, France.  
OUGH, S. W.  
PARKS, HENRY M., Captain, Engineer Officers' Reserve Corps.  
PERRY, O. B., Captain, Engineer Officers' Reserve Corps.  
PICKETT, CHAS. E., B Battery, 5th Field Artillery, U. S. Army.

PONTON, G. M., Lieutenant C. E. Explosives Dept., Imperial Munitions Board, Ottawa, Ont., Canada.

PULLEN, ERNEST F.

REECE, FRED B.

REYNOLDS, GEORGE B.

ROCKWELL, F. Y., Officers' Reserve Corps, Fort Myer, Va.

RODGERS, ALAN M., 2d Lieutenant, Engineer Officers' Reserve Corps.

ROOT, JOHN A., Captain, Ordnance Dept., Engineer Officers' Reserve Corps.

ROSS, JAMES G., Captain, Discharged from active service.

ROUDABUSH, M. A., 2d Lieutenant, Engineer Officers' Reserve Corps.

SACKET, CHARLES T., Captain, Engineer Officers' Reserve Corps.

SANDERS, BERNARD H., 254th Engineers, France.

SCOTT, JOHN T., 9th Co., 9th Division, Ft. Benjamin Harrison.

SCOUTLAR, A. C., Major, 5th Border Regiment of South Africa, France.

SEMPLE, R. A., Sapper, Canadian Engineers.

SIBBALD, ALEXANDER.

SMULLEN, C. KENNETH, 2d Lieutenant, Engineer Officers' Reserve Corps.

SPURNY, EMIL, U. S. Army Hospital No. 2, London War Office.

STEHLI, H. J., Captain, Engineer Officers' Reserve Corps.

STEVEN, H. M., Lieutenant.

TERRY, MARK L., 3d Battery, Field Artillery, Ft. Riley, Kans.

VALLET, BENJAMIN W., Major, Battalion of Engineers, Michigan College of Mines.

VAN NESS, W. WATERS, Captain, Royal Engineers, British Army.

WATKINS, JOSEPH C., Captain, Engineer Officers' Reserve Corps.

WEBB, TORREY H., Signal Service Reserve Corps.

WEIGELL, HENRY S., Private, Artists' Rifles Platoon 9-C. Co., B. E. F.

WEINBERG, GEORGE S., Major, Engineer Officers' Reserve Corps.

WILKINSON, W. F., Captain.

WORTH, JOHN G., Captain Quartermaster, Officers' Reserve Corps.

WRAITH, C. R., Captain, Ordnance Dept., U. S. R.

YOUNG, HORACE G.

**Request for Information.**—We would appreciate information concerning the official position of members whose names are printed in the above list without title (who are now in active service at the front), and also concerning other members who are engaged in any military service.



## PERSONAL

The following is a partial list of members and guests who called at Institute headquarters during the period July 10, 1917, to Aug. 10, 1917:

|  |                                     |
|--|-------------------------------------|
| G. A. Collins, Seattle, Wash.            | Norman Picot, Melbourne, Aust.      |
| A. D. R. Galloway, Komfrodud, W. Africa. | W. E. Pomeroy, El Paso, Tex.        |
| F. Lynwood Garrison, Philadelphia, Pa.   | Francis R. Pyne, Elizabeth, N. J.   |
| Geo. M. Henderson, Bland, N. M.          | W. L. Remick, W. Norfolk, Va.       |
| L. B. Herr, Jr., So. Bethlehem, Pa.      | M. F. Sayre, Schenectady, N. Y.     |
| Zay Jeffries, Cleveland, O.              | Leighton Stewart, New York, N. Y.   |
| H. H. Lauer, Allentown, Pa.              | Randolph D. Talmage, Joplin, Mo.    |
| Wm. B. McKinlay, Yonkers, N. Y.          | H. J. Volker, Ossining, N. Y.       |
| A. N. Mackay, Republic of Colombia.      | Dwight E. Woodbridge, Duluth, Minn. |
| Edmund Newton, Minneapolis, Minn.        | Kwong Wu, Peking, China.            |

J. N. Acosta has resigned his position as mining engineer for the Shannon Copper Co., Gleeson, Ariz., and is now engaged as consulting engineer for Lower California, Mexico, with headquarters at San Diego, Cal.

L. D. Albin, formerly assistant general sales manager of the Ingersoll-Rand Co., was appointed general sales manager on July 25, 1917.

R. A. Bull, for the past year connected with the Chicago Steel Foundry Co., has been elected vice president and general manager of the Duquesne Steel Foundry Co., Coraopolis, Penn.

S. H. Chauvenet, formerly with the Robeson and Sheridan furnaces in eastern Pennsylvania, will act as manager at the Island Park furnace, Easton, Pa., recently purchased by the Northern Ore Co.

John F. Cregan is now in the employ of The American Zinc and Chemical Co., Langloeth, Pa.

C. L. Dake, Assistant Professor of Geology of the Missouri School of Mines, has been granted one year's leave of absence. During this time he will do field work in Oklahoma for Valerius, McNutt & Hughes.

J. C. Dick and C. T. Van Winkle announce that they have opened offices for the practice of mining and metallurgical engineering at 505-6 Scott Bldg., Salt Lake City, Utah.

M. M. Duncan, vice president and general manager of the Cleveland-Cliffs Iron Co., has been reappointed a member of the board of control of the Michigan College of Mines, by Governor Sleeper. Mr. Duncan has served on the board for several terms.

Charles R. Fettke is engaged in making a geological survey of Clay and Macon Counties, Tenn., with particular reference to the possible occurrence of oil and gas in this area.

**Francis N. Flynn** has accepted a position with the Consolidated Mining & Smelting Co., of Canada, Trail, B. C.

**A. O. Gates**, Salt Lake sales engineer for the Dodge Sales & Engineering Co., has been ordered to active duty as lieutenant in the United States Naval Reserve.

**Herbert W. Gepp** has been appointed general manager of the Electrolytic Zinc Co., of Australia, Prop.

**Bancroft Gore** of the Department of Metallurgy, Montana State School of Mines, Butte, Mont., sailed from New York on Aug. 10 for El Monte, Prov. of Santiago, Chile, where he is to have charge of the new powdered coal reverberatory installation of the Société des Mines de Cuivre de Naltagua.

**Otto Herres, Jr.**, has accepted a position as superintendent of the Hiawatha Mine, United States Fuel Co., Hiawatha, Utah.

**J. F. Hoffman** has severed his connection with the Nevada Douglas Consolidated Copper Co., to accept a position with the Empire Steel & Iron Co., Mt. Hope, N. J.

**Scovill E. Hollister** has accepted a position as resident manager, The Leviathan Mines Co., Yucca, Ariz.

**J. H. Jowett**, formerly general sales manager of the Ingersoll-Rand Co., was elected vice-president of the company on July 25, 1917.

**John G. Kirchen**, general manager of the Tonopah Extension Mining Co., and allied interests, will remove to Reno, Nev. He will retain his position with the company

**Warren Lawson Klutz**, who has had an extensive experience in the operation of blast furnaces in the South, has been appointed general manager of the Sheffield Coal & Iron Co., Sheffield, Ala.

**Isak Partenan** has accepted a position with the Liberty Bell Gold Mining Co., Telluride, Colo.

**C. Thomas Patterson** has accepted a position as metallurgist for the Standard Chemical Co., Houston, Pa.

**Harold Rabling** has resigned his position with the St. Joseph Lead Co., Bonne Terre, Mo., and has sailed for Australia to enlist in the Australian Forces.

**Stuart Rawlings** arrived in Peru last month to take the position of assistant general manager of the Cerro de Pasco Mining Company.

**Forest Rutherford**, for many years superintendent of reduction works for the Copper Queen Consolidated Mining Co., Douglas, Ariz., resigned his position on June 1, and will go into business for himself as a consulting metallurgical engineer, First National Bank, Pueblo, Colo.

**Carl Scholz**, manager, Mining & Fuel Department of The Chicago, Rock Island & Pacific Railway Co., has resigned to accept service with the Chicago, Burlington & Quincy R. R. Co.

**C. L. Severy** has severed his connection with the Carter Oil Co. and is doing consulting geological work under the firm name of Pemberton & Severy.

**H. J. Stehli**, Captain in the First Engineer Regiment, has been assigned to Colonel McKinstry's staff, and is now at Fort Totten.

**David Taylor**, president of the Consolidated Ores Co., has closed his office in the Boston Building, Salt Lake City, and is now at Symes Building, Denver, Colo.

**Arthur C. Terrill** is spending several weeks in the new Kansas-Oklahoma Zinc Field gathering data for the State Geological Survey of Kansas.

**Arthur P. Watt**, metallurgist for the St. Louis Smelting & Refining Co., has resigned his position and is now metallurgist for the Missouri Metals Corp., at Mine La Motte, Mo.

**E. W. Westervelt** has been appointed superintendent of the Hobson Silver-Lead Co., at Ymir, British Columbia.

**A. R. Weigall** has been appointed general manager for the Seoul Mining Co., in Korea.

**Frank R. Wicks** has succeeded W. H. Janney as manager of the mill of Chino Copper Co., at Hurley, N. M.

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### POSITIONS VACANT

No. 230. Two engineers, technical graduates, three to five years, experience. One to take charge of pyrite mining in Canada; the other to assist in the mining work at the head office in New York. Salary for the right man, \$5000.

No. 231. Two young engineers, preferably men who have just graduated from college who are looking for an opportunity to work up with a reputable mining company. Men who have had some practical experience will be considered.

No. 232. Assayer. Wet analyses, principally of lead, zinc, sulphur and insoluble. Assayer will also have some mill sampling and experimental work in a new flotation installation, as well as control of regular mill operation and development-work samples from the mine. \$75 to \$100 per month and board, according to experience of candidate.

No. 233. Engineer to lay out, erect and operate calcium carbide and ferrosilicon plant in the State of Washington. One furnace, 1500 hp.

No. 234. Professor of metallurgy for school of mines in western state. \$3000 per year to man properly qualified for the work.

No. 235. Permanent position in Siberia for capable engineer experienced in flotation and table work. Excellent opening for right man. Living conditions good for family.

No 236. Position open for transitman and draftsman, salary about \$100 per month and expenses in the field. Headquarters, St. Louis.

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### ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, aged 31. Five years' experience in zinc smelting, as chief chemist and superintendent; two years foreman and superintendent zinc refinery. Thorough assayer and chemist, both in theory and practical experience. Experienced cement chemist. Married. Reference. At present employed. No. 358.

Concentrator superintendent. Just returned from abroad. Open for engagement. Exceptional experience designing, construction and operation. References A1. Twenty years' experience. No. 362.

Mine manager, member, technical graduate with many years' experience, desires change. Now successfully managing producing mine in the Southwest. No. 372.

Mining engineer and executive. Member. Columbia graduate. Age, 32. Married. 9 years' responsible work in mine examination and development. Experience in United States, Canada, Mexico and South America. Now superintendent large operating mine. Personal interview in New York. Available now. No. 367.

Member, technical graduate, ten years in silver mines and mills of Nevada; superintendent of well-known mine and mill; desires change; specialty, mining and cyanidation of silver ores; student of cost and business organization; best of references from prominent engineers. No. 374.

Member. Age, 27. Married. Technical graduate. Experience in Wisconsin and Missouri zinc fields. Desires work as superintendent or on mechanical engineering staff. No objection to location. Available after August 15. No. 375.

Member desires position as mining geologist—management, investigation, research. Expert in surface and underground geological surveys, direction of diamond drilling and exploration, mineralogical analysis and methods of milling, development of mining properties and handling of difficult mining and milling problems. No. 376.

Member. Age, 30. Experience as mine sampler, solution man in mill, assistant in refinery, helper and runner diamond drill, superintendent of mine. No. 377.

Member, 34 years old, desires position constructing and operating sulphur dioxide plant, preferably on the Pacific Coast. Has had experience in experimental mill work, mill construction and operation,

underground surveying, construction and operation of plant for manufacture of anhydrous liquid sulphur dioxide. No. 378.

Blast-furnace superintendent or manager. Member who has been general superintendent and general manager in blast furnaces, iron ore mines, coal and coke operations, stone quarries, etc. No. 379.

Member, who speaks Spanish and German, wants position as general superintendent or manager. Has been cyanide superintendent, general superintendent, general manager, and has made various mine examinations and metallurgical investigations, including amalgamation concentration, cyanidation, and flotation. No. 380.

Member, married, 20 years' experience. Assayer and surveyor, afterwards superintendent of silver and gold mines. No. 381.

Man who has had experience in cyanide work, and as machine man, transit man and head assayer, desires position with manufacturer of mining or milling machinery or with chemical company in or near New York City. No. 382.

Mining engineer, graduate Columbia, experienced surveying, time-keeping, supply purchasing, safety work, mining and construction jobs, desires change. Married. Age, 30. \$2100. Month's notice. No. 383.

## LOCAL SECTION NEWS

### NEW YORK SECTION

J. E. JOHNSON, JR., *Chairman*      EDGAR RICKARD, *Vice Chairman*

D. M. LIDDELL, *Secretary*, 7 Wall St., New York, N. Y.

C. A. BOHN, *Treasurer*

JOHN V. N. DORR

LEWIS W. FRANCIS

The following minute was unanimously passed  
at a meeting of the

NEW YORK SECTION

AMERICAN INSTITUTE OF MINING ENGINEERS

held on the  
evening of June 6, 1917,

two regular meetings having been adjourned by unanimous consent until this date out of respect to the memory of the late Chairman of the Section.

#### *Minute*

The death of our beloved fellow member

DR. DAVID H. BROWNE

one of the foremost metallurgists and one of the most brilliant and original thinkers in the profession of mining engineers, the chairman for the past two years of this section, has caused throughout a wide circle of

friends, colleagues and admirers a common feeling of great sorrow. The vast respect felt by his colleagues for his achievements in the science and art of metallurgy, was second only to the affection felt for him as a friend. Under his leadership the New York Section of the American Institute of Mining Engineers made great progress. His inspiration was felt by all, but the younger men of the profession especially were helped by his interest in their welfare and his thoughtful plans for their entertainment and their betterment throughout the meetings of the Section.

Mourning with a multitude of friends his departure and his premature death, the Section shares with them a feeling of pride and gratitude for his comradeship and his work among us.

The secretary of the Section is instructed to transmit a copy of this minute to Dr. Browne's family with assurances of the Section's sympathy and participation in its loss.

Committee { J. E. JOHNSON, JR.,  
BRADLEY STOUGHTON,  
ALBERT D. BEERS.

On July 20, a special war meeting of the Section was held at which the subject of the supply of manganese was discussed. The report of this meeting will be found in the preceding pages of this *Bulletin*.

### SOUTHERN CALIFORNIA SECTION

A. B. W. HODGES, *Chairman*, RALPH ARNOLD, *Vice-Chairman*

ALVIN B. CARPENTER, *Secretary-Treasurer*, 530 Citizens National Bank Building, Los Angeles, Cal.

H. KENYON BURCH

W. F. STAUNTON

C. COLCOCK JONES

W. H. WILEY

The Annual Meeting of the Southern California Section was held at the Sierra Madre Club on May 5, 1917, and began with a dinner.

Forty seven members were present. The guests of the evening were Mr. Philip N. Moore, President of the Institute and Mr. Bradley Stoughton, Secretary. Mr. Robert Sticht, formerly president of the Australian Institute of Mining Engineers, was also a guest of the Section.

After the dinner, President Moore gave a most interesting talk on the activities of the Institute and on what can be done towards increasing the influence of the Institute and enlarging its activities. He urged the enrolling of all mining engineers that are qualified, and hoped the time would come when the public and the layman would demand membership in the Institute as an evidence of an engineer's ability and character. After discussion of the main points presented by Mr. Moore, Mr. Stoughton spoke of "secretaryships" and their relation to the Sections. He also brought out many important phases of the Institute's activities, and aroused interested discussion on the subject of the present usefulness of the Bulletin. The idea of making it more of a magazine and including an advertising department devoted to mining and metallurgical equipment was presented and discussed.

The general sentiment of the members present was very much in favor of the magazine idea, along the lines of the magazines published by the

American Society of Mechanical Engineers and the American Medical Society, with a compilation of an annual index of advertisers.

The business of the Annual Meeting was then taken up and, after the reading and adoption of the reports of the Secretary-Treasurer, the result of the ballot for the officers for the ensuing year was read. The names of the officers elected appear at the beginning of this article.

The meeting then adjourned.

ALVIN B. CARPENTER, *Secretary.*

### FORTHCOMING MEETINGS OF SOCIETIES

| Organisation                                     | Place             | Date<br>1917 |
|--|-------------------|--------------|
| National Association of Stationary Engineers.... | Evansville, Ind.  | Sept. 10-15  |
| National Safety Council.....                     | New York City.    | Sept. 10-15  |
| National Exposition of Safety and Sanitation.... | New York City.    | Sept. 10-15  |
| American Iron, Steel & Electrical Engineers..... | Philadelphia, Pa. | Sept. 10-14  |
| American Association of Port Authorities.....    | Cleveland, Ohio.  | Sept. 11-14  |
| American Institute of Metals.....                | Boston, Mass.     | Sept. 24-29  |
| American Foundrymen's Association.....           | Boston, Mass.     | Sept. 24-29  |
| Chemical Industries, Third National Exhibition.. | New York City.    | Sept. 24-29  |
| American Electrochemical Society.....            | Pittsburgh, Pa.   | Oct. 3-6     |
| American Institute of Mining Engineers.....      | St. Louis, Mo.    | Oct. 8-13    |
| American Gas Institute.....                      | Washington, D. C. | Oct. 16-19   |
| American Institute of Architects.....            | Philadelphia, Pa. | Dec. 26-29   |
| American Society of Mechanical Engineers.....    | New York City.    | Dec. 26-29   |
| American Mining Congress State Chapter.....      | Phoenix, Ariz.    | Dec. 26-29   |

## LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.

AMERICAN SOCIETY OF MECHANICAL ENGINEERS.

AMERICAN INSTITUTE OF MINING ENGINEERS.

UNITED ENGINEERING SOCIETY

· HARRISON W. CRAVER, Director

The Library of the above-named Societies is open from 9 A. M. to 10 P. M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

### LIBRARY ACCESSIONS

#### PARTIAL LIST CLASSIFIED BY SUBJECTS

##### Mining and Metallurgy

- UNITED STATES BUREAU OF MINES EXPERIMENT STATION AT TUCSON. (University of Arizona. Bulletin 7.) Tucson, 1915-16. (Gift of Arizona Bureau of Mines.)
- BIBLIOGRAPHY OF ARIZONA MINING, METALLURGY AND GEOLOGY. (University of Arizona. Bulletin No. 23.) Tucson, 1915-16. (Gift of Arizona Bureau of Mines.)
- "DINAS" ESTRATTO DALLA RIVISTA. "La Metallurgia Italiana," 31 gennaio 1917, n.1. By F. Bondolfi. Milano, 1917. (Gift of author.)
- DIRECTORY OF ARIZONA OPERATING MINES. (University of Arizona. Bulletin 24.) Tucson, 1915-16. (Gift of Arizona Bureau of Mines.)
- EXTRALATERAL RIGHT, SHALL IT BE ABOLISHED? By Wm. E. Colby. (Reprint of a series of articles published in the California Law Review, 1916-17.) (Gift of author.)
- IRON. (University of Arizona. Bulletin 43.) Tucson, 1916-17. (Gift of Arizona Bureau of Mines.)
- MILL AND SMELTER METHODS OF SAMPLING. (University of Arizona. Bulletin No. 26.) Tucson, 1915-16. (Gift of Arizona Bureau of Mines.)
- ORGANIZING AND CONDUCTING SAFETY WORK IN MINES. (U. S. Bureau of Mines. Technical Paper 103.) Washington, 1917.
- STATE SAFETY NEWS, June, 1916. (University of Arizona. Bulletin No. 30.) Tucson, 1915-16. (Gift of Arizona Bureau of Mines.)
- August, 1916. (University of Arizona. Bulletin No. 32.) Tucson, 1916-17. (Gift of Arizona Bureau of Mines.)



## Geology and Mineral Resources

- CONTRIBUTIONS TO ECONOMIC GEOLOGY, 1916. PART II—MINERAL FUELS. (U. S. Geological Survey. Bulletin No. 641.) Washington, 1917.
- GEMS AND PRECIOUS STONES OF ARIZONA. (University of Arizona. Bulletin 48.) Tucson, 1916-17. (Gift of Arizona Bureau of Mines.)
- GEOLOGY AND MINERAL RESOURCES OF THE REEFTON SUBDIVISION, WESTPORT AND NORTH WESTLAND DIVISIONS. (New Zealand. Geological Survey Bulletin No. 18.) Wellington, 1917.
- GEOLOGY AND MINING INTERESTS OF THE BLACK HILLS REGION, BIBLIOGRAPHY OF. (South Dakota. School of Mines. Dept. of Geology. Bulletin No. 11.) Rapid City, 1917. (Gift of South Dakota School of Mines.)
- GEOLOGY OF A PORTION OF THE FLATHEAD COAL AREA, BRITISH COLUMBIA. (Canada, Mines Department. Memoir 87.) Ottawa, 1916.
- MINERALOGY OF USEFUL MINERALS IN ARIZONA. (University of Arizona. Bulletin No. 41.) Tucson, 1916-17. (Gift of Arizona Bureau of Mines.)
- MISCELLANEOUS MINERALS. (University of Arizona. Bulletin 49.) Tucson, 1916-17. (Gift of Arizona Bureau of Mines.)
- NORWAY. Statistisk aarbok for kongeriket Norge, 36th aargang, 1916. Kristiania, 1917. (Gift of Norge Statistiske Centralbureau.)
- ONTARIO BUREAU OF MINES. 25th Annual Report, 1916. Toronto, 1916. (Gift of Ontario Bureau of Mines.)
- PENNSYLVANIA GLACIATION FIRST PHASE. By Edw. H. Williams, Jr. Woodstock, Vt., 1917.
- RIVISTA DEL SERVIZIO MINERARIO NEL 1915. Roma, 1917. (Gift of Ministero d'Agricoltura, Rome, Italy.)

## Chemistry

- MANUFACTURE OF SULPHURIC ACID AND ALKALI WITH THE COLLATERAL BRANCHES, Ed. 4. Supplement to volume I. Sulphuric and Nitric Acid. By George Lunge. London, 1917.

## General

- AMERICAN RAILWAY ENGINEERING ASSOCIATION. Manual, 1915. (Gift of Association.)
- BRITISH COLUMBIA. Water Rights Branch. Report, 1916. Victoria, 1917. (Gift of Water Rights Branch.)
- CEMENT. (University of Arizona. Bulletin No. 25.) Tucson, 1915-16. (Gift of Arizona Bureau of Mines.)
- CEMENT WORLD, vols. 1-2. Chicago, 1907-08.
- PENNSYLVANIA. COMMISSIONER OF LABOR AND INDUSTRY. 2d Annual Report, 1915, Part II. Harrisburg, 1916.
- TEXT BOOK ON MOTOR CAR ENGINEERING. Volume II—Design. By A. G. Clark. London, 1917.
- THE WESTERN'S STEEL BOOK, A HANDY REFERENCE BOOK FOR THE AUTOMOBILE ENGINEER. Edited by Erwin C. Arndts. Cincinnati, 1917. (Gift of The Western Spring & Axle Co.)
- UNDERWRITERS' LABORATORIES. List of Inspected Mechanical Appliances, July, 1916. List of Inspected Electrical Appliances, April, 1917.
- VALUE OF PEAT FUEL FOR THE GENERATION OF STEAM. (Canada. Mines Department. Bulletin 17.) Ottawa, 1917.
- VELADA QUE EN HONOR DEL SEÑOR INGENIERO DE MINAS, D. Manuel Maria Contreras. Mexico, 1902.

## Trade Catalogs

- INGERSOLL-RAND COMPANY. New York City.  
Form No. 3118. Compressors and Vacuum pumps for extraction of gasoline from natural gas. June, 1917.  
— 4302. "Sergeant" rock drills. May, 1917.
- NATIONAL STEAM PUMP COMPANY. Upper Sandusky, Ohio.  
Bulletin No. 5. National Hydraulic Pressure Pump.  
— No. 20. National Boiler Feed Pump, balanced piston valve.  
— No. 21. National outside packed plunger pump, balanced piston valve. Class B.  
— No. 22. National Power Pumps.  
— No. 23. National Duplex Pumps.  
— No. 24. National Oil Line Pump.

- Illustrated Catalog No. 29. Pumping Machinery and Air Compressors.  
 National Deep Well Pumping Engine.  
 National Power Pumps.  
 National Pumps.  
 National Wet Vacuum Pumps (Special pattern).  
 THEW AUTOMATIC SHOVEL COMPANY. Lorain, Ohio.  
 Catalog No. 10. Thew shovels, steam, electric and gasoline.  
 WRIGHT WIRE COMPANY. Worcester, Mass.  
 General catalog. 1912.

### Book Notices

Unless otherwise specified, books in the following list have been donated by the publishers. The Institute does not assume responsibility for any statements made. They are taken either from the preface or the text of the book.

**COMPRESSED AIR. THEORY AND COMPUTATIONS.** By Elmo G. Harris. 2d ed., rev. and enl. N. Y., McGraw-Hill Book Co., Inc.; Lond., Hill Publishing Co., Ltd. 1917. 15 + 192 pp., 32 illus., 9 × 6 in., cloth, \$2.

Designed to present the mathematical treatment of the problems in the production and application of compressed air. In the present edition errors and ambiguities have been eliminated and new matter added.

**OFFICE ORGANIZATION AND MANAGEMENT.** By Carl C. Parsons. Chic., LaSalle Extension University, 1917. 14 + 313 pp., 59 illus., 1 diagram, 8 × 6 in., leather, \$2.50.

Treats of organization, management, layout, equipment, methods, systems, records, forms, employees, etc. Based on observation of the methods used in the offices of various large companies.

**INTRODUCTION TO THE RARER ELEMENTS.** By Philip E. Browning. 4th ed. rev. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1917. 10 + 250 pp., 6 illus., 1 pl., 9 × 6 in., cloth, \$2.

The first edition of the book was prepared by the author from material used in a short lecture course at Yale University, as a convenient handbook in the introductory study of the rarer elements. Numerous changes and additions have been made in the present edition, in order to bring it up to date.

**GAS AND FUEL ANALYSIS FOR ENGINEERS.** A Compend for Those Interested in the Economical Application of Fuel. Prepared Especially for the Use of Students at the Massachusetts Institute of Technology. By Augustus H. Gill, 8th ed. rev. N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1917. 145 pp., 20 illus., 7 × 5 in., cloth, \$1.25.

An attempt to present in a concise yet clear form the methods of gas and fuel analysis involved in testing the efficiency of a boiler plant. The directions for using the bomb calorimeter have been made to agree more closely with present usage in this edition, and minor corrections and additions have been made throughout the text.

**CHEMICAL DISCOVERY AND INVENTION IN THE TWENTIETH CENTURY.** By Sir William A. Tilden, N. Y., E. P. Dutton & Co.; Lond., George Routledge and Sons, Ltd. (preface 1916). 16 + 487 pp., 150 illus., 11 por., 9 × 6 in., cloth, \$3.50.

A semi-popular account of modern chemical discovery, covering both theory and applications. Contents: Chemical Laboratories and the Work Done in Them; Modern Discoveries and Theories; Modern Applications of Chemistry; Modern Progress in Organic Chemistry.

**PAINT RESEARCHES AND THEIR PRACTICAL APPLICATION.** By Henry A. Gardner. Washington, D. C. (privately printed) 1917. 384 pp., 155 illus., 9 × 6 in., cloth, \$5.

This volume is a summary of the author's investigations conducted at the Institute of Industrial Research for the Educational Bureau of the Paint Manufacturers' Association of the United States. Contents: The Growth of the Prepared Paint Industry and Its Relation to the Work of the Painter; The White Pigment Industry; Physical Characteristics of Pigments and Paints; Tests of Lithopone; Washington Paint Oil Tests; Paint Protection for Portland-Cement Surfaces; Paints to Prevent Electrolysis in Concrete Structures; Paints for Metal; Marine Paints; Arlington Paint Tests; Observations on Painted Lumber; Impregnated Panel Tests; Fire Retardant Paints for Shingles and Other Wooden Structures; The Composition of Paint Vapors; The Toxic and Antiseptic Properties of Paints; The Light-Reflecting

Values of White and Colored Paints; Formation and Inhibition of Mildew in Paints; Fungi on Painted Surfaces; Changes Occurring in Oils and Paste Paints, Due to Autohydrolysis of the Glycerides; The Effect of Pigments upon the Constants of Linseed Oil; Storage Changes in Vegetable and Animal Oils; Paint Driers and Their Application; Miscellaneous Oil Investigations; The Application of Paints and Finishes to Wood.

**STRENGTH OF MATERIAL.** By James E. Boyd. 2d ed. rev. and enl., N. Y., McGraw-Hill Book Co., Inc.; Lond., Hill Publishing Co., Ltd., 1917. 16 + 380 pp., 254 illus., 9 × 6 in., cloth, \$3.

No radical changes have been made in the present edition of this book. A chapter on Curved Beams and Hooks has been added and part of a chapter on Theories of Failure. The most important addition is the method of Area Moments for deriving the equations of the elastic line of beam. The book has been designed to be used in connection with either "Cambria Steel" or "Carnegie Pocket Companion."

**NAVAL ARCHITECTURE.** By Cecil H. Peabody. 4th ed., rev., N. Y., John Wiley & Sons, Inc.; Lond., Chapman & Hall, Ltd., 1917. 641 pp., 218 illus., 1 diagram, 9 × 6 in., cloth, \$7.50.

Intended to give a consistent and connected statement of the commonly accepted theory of naval architecture. Intended for naval architects and shipbuilders, as well as for students.

Discussions of certain devices for checking rolling of ships, such as Frahm's anti-rolling tanks and the steadying gyroscopes of Schlick and Sperry, have been introduced in this edition.

**SHIPYARD PRACTICE AS APPLIED TO WARSHIP CONSTRUCTION.** By Neil J. McDermaid. 2d ed. N. Y., and Lond., Longmans, Green & Co., 1917. 332 pp., 153 illus., 9 × 6 in., cloth, \$4.

A course of lectures given to Cadets of Naval Instruction at the Royal Naval College, Devonport. Describes the actual operations to be performed during the construction and outfitting of a warship.

**WAR-SHIPS.** A Text-Book on the Construction, Protection, Stability, Turning, Etc. of War Vessels. By Edward L. Attwood, 6th ed., N. Y. and Lond., Longmans, Green and Co., 1917. 338 pp., 209 illus., 6 × 9 in., cloth, \$4.

This book has been written in response to suggestions made by senior naval officers taking the course in Naval Architecture at the Royal Naval College, Greenwich. An attempt has been made to treat the subject from the naval officers' standpoint and certain parts have been treated with a view to meeting their special requirements.

**UNITED STATES ARTILLERY AMMUNITION.** 3 to 6 in. Shrapnel Shells, 3 to 6 in. High Explosive Shells and Their Cartridge Cases. By Ethan Viall. N. Y., McGraw-Hill Book Co., Inc.; Lond., Hill Publishing Co., Ltd., 1917. 98 pp., 171 illus., 12 × 9 in., cloth, \$2.

This work is intended to give shop men, engineers and manufacturers an accurate knowledge of the sizes, tools, shop work and gages for the more commonly used United States shells and cartridge cases. The descriptions are in minute detail and accompanied by numerous dimensioned drawings.

**EXPORT TRADE DIRECTORY 1917-1918.** Compiled by B. Olney Hough. N. Y., American Exporter, Johnston Export Publishing Co. 537 pp., 1 map, 9 × 6 in., cloth, \$5.

Contents: Export Merchants in the United States; Manufacturers' Export Agents, Managers of Export Departments and Export Brokers; Leading Bankers Engaged in Foreign Exchange Business; Foreign Exchange Brokers; Marine Insurance Companies in New York City; Foreign Freight Forwarders; Some Export Trucking Companies in New York City; Steamship Services to Foreign Ports; How to Ship to Foreign Markets; Consuls of Foreign Countries in the United States; United States Consular and Commercial Representatives in Foreign Countries; Associations for the Promotion of Export Trade.

**MACRAE'S BLUE BOOK.** America's Greatest Buying Guide. Vol. 8, 1917. Chic. and New York, MacRae's Blue Book Co., 1344 pp., 11 × 9 in., cloth, \$10.

Contents: Catalogue Section; Address Section; Representatives' Index; Classified Material Section; Comprehensive Trade Name Index; Miscellaneous Data Section; Standard List Prices of Building Materials and Iron and Steel Products; A Net Discount Computer.

## MEMBERSHIP

## NEW MEMBERS

The following list comprises the names of those persons who became members during the period of July 10, 1917, to Aug. 10, 1917.

- ARNOT, STANLEY L., Engr., Plymouth Cons. Gold Mines, Ltd.,  
Plymouth, Amador Co., Cal.
- BARRETT, LESLIE PARK, Asst. Geol.... Michigan Geological Survey, Lansing, Mich.
- BENNIE, JOHN WILLIS.....Gen'l. Mgr., Shannon Copper Co., Clifton, Ariz.
- BRINTON, OWEN FUGATE.....Mgr., Admiralty Zinc Co., Quapaw, Okla.
- BRODERMANN, JORGE, Civil Engr., Compania Minera San Francisco, Havana, Cuba.
- BROWN, RICHARD HENRY JR., Sales Engr., Ingersoll Rand Co.,  
300 N. Broadway, St. Louis, Mo.
- BROWN, STANLEY.....Chile Exploration Co., Chuquicamata, Chile.
- BUDDECKE, WILLIAM A.....Pres., Point Milling & Mfg., Co. Mineral Point, Mo.
- BURGESS, GEORGE K., Physicist and Chief, Division of Metallurgy,  
Bureau of Standards, Washington, D. C.
- BURNHAM, A. J., Gen'l. Mgr., Chicago Mines Co., 303 Miners Bank Bldg., Joplin, Mo.
- CACERAS, FEDERICO GARCIA, Met.; Chem., Negociacion Minera Fernandini,  
Oficina Huaracaca, La Fundicion, Peru, S. A.
- CHAPMAN, GEORGE ALBERT, Met., Minerals Separation North American Corp.,  
61 Broadway, New York, N. Y.
- CHAPMAN, JAMES, Mech. Engr., Fraser & Chalmers Ltd.,  
5th Floor Corner House, Johannesburg, Transvaal.
- CLARKSON, JOHN L.....Gen'l. Mgr., Nashville Mining Co., Nashville, Ill.
- DAKE, CHARLES LAWRENCE, Asst. Prof. of Geology, School of Mines & Met., Rolla, Mo.
- DALBEY, GEORGE E....Met., American Smelting & Refining Co., Omaha, Neb.
- DAY, FRANKLIN KAERCHER, Chief Engr., W. Va. Div.,  
The Consolidation Coal Co., Fairmont, W. Va.
- DIETZ, JOHN HALIFAX, Cons. Engr.....866 Rockefeller Bldg., Cleveland, O.
- DOVE, DEAN REID, Chief Chem., Magna Plant, Utah Copper Co.,  
Box 42, Magna, Utah.
- ENEBO, BENJAMIN A., Chief Draftsman & Supt. of Constr., Chino Copper Co.,  
Hurley, New Mex.
- FEILD, ALEXANDER L., Asst. Met., U. S. Bureau of Mines, Univ. of Minnesota,  
Minneapolis, Minn.
- FRANCE, DE WAYNE L.....Met. Engr., Scovill Mfg. Co., Waterbury, Conn.
- GAY, HAROLD SINCLAIR.....200 S. Wilson Ave., Alhambra, Cal.
- GILLIS, JULIUS H., Chief Draftsman, British America Nickel Corp., Ltd.,  
507 Royal Bank Bldg., Toronto, Ont. Can.
- GRINDLAY, L. B., Inspecting Engr., Republic Iron & Steel Co., Youngstown, O.
- HAHN, ALBERT W., Met. Mgr., Electrolytic Zinc Co. Inc., & Metals Disintegrating Co.  
Inc., 3 South William St., New York, N. Y.
- HARRIS, JOHN S., Asst. Mill Supt., American Zinc Lead & Smelting Co., Granby, Mo.
- HECKEL, FRED J.....Vice-Pres., Heckel Oil Co., 47 William St., Bradford, Pa.
- JARVIS, JOSEPH R.....Mgr., Hay Fork Mines, Idaho City, Idaho.
- KENNEDY, GERALD JOSEPH, Gen'l. Supt., Bethlehem Chile Iron Mines Co., Tofo,  
La Higuera, Coquimbo, Chile.
- LAVERY, JOHN CLYDE.....Met., Chino Copper Co., Hurley, New Mex.
- LEHMAN, ANDREW WILLIAM, Min. Engr., Gen'l. Mgr., Société des Mines de Cuivre de  
Catémou, El Nilhue, Chagres, near Valparaiso, Chile.
- McFARLAND, HORACE M.....623 1st Natl. Bank Bldg., Birmingham, Ala.
- MITCHELL, JAMES MACDONALD, Supt., St. Nicholas Zinc Co., Summitville, N. Y.
- MORSE, CHARLES WOODMAN, Concentrating Engr., Washoe Reduction Works,  
Anaconda Copper Min. Co., Anaconda, Mont.
- MURK, NEAL M., Min. Engr.....1024 Mills Bldg., San Francisco, Cal.
- OAA8, TORGUS HANS, Foreman, High Ore Mine, Anaconda Copper Mining Co.,  
Butte, Mont.

- O'ROURKE, JAMES EUGENE, Supt. of Constr., Potter Construction Co.,  
Railway Exchange Bldg., St. Louis, Mo.
- OWEN, ALBERT LEE, Vice-Pres. & Supt. Royal John Min. Co., Santa Rita, New Mex.
- PEMBERTON, JOHN ROTHWELL, Geol., Pemberton & Severy, Cons. Geol.,  
802 Starks Bldg., Louisville, Ky.
- PERRY, RALPH G. .... Engineering Dept., Ray Cons. Copper Co., Ray, Ariz.
- PERRY, WILLIAM A., Gen'l. Mgr., Greensburg-Connellsville Coal & Coke Co.,  
2410 First National Bank Bldg., Pittsburgh, Pa.
- PFABLER, FRED S., Gen'l. Supt., Chicago & Northwestern Coal Properties,  
Gillespie, Ill.
- PONTON, GERALD MUNGO ..... Sidney Cottage, Belleville, Ontario, Can.
- POWERS, SIDNEY, Pet. Geol. .... 409 Second Ave., Troy, N. Y.
- RICHTER, EINAR CHRISTIAN, Mgr., Luossavaara-Kiirunavaara Aktiebolag,  
Narvik, Norway.
- RISER, ORSON FRITZ, Asst. Supt. of Mills, Chino Copper Co., Hurley, New Mex.
- SCHUBERT, ROBERT, Met. Engr., Stavanger Electro Staalverk A/S, Stavanger, Norway.
- SHACKELFORD, WARREN BARTON, Gen'l. Mgr., Aiken, Curtis & Cosden,  
Webb City, Mo.
- SNYDER, BLANCHARD M., Gen'l. Mgr., Arizona Middlemarch Copper Co., Pearce, Ariz.
- SPEED, FREDERIC R. .... Beechwood Ave., Catonsville, Md.
- SULBERG, KARL. .... Gen'l. Supt., United States Smelting Co., Checotah, Okla.
- TOWNE, JOHN LEO ..... Supt., Akron Mines, Whitepine, Colo.
- VAN WAGENEN, HUGH R. .... Care Allen H. Rogers, 71 Broadway, New York, N. Y.
- WADHAMS, ALBION JOHN, Asst. Gen'l. Supt., The International Nickel Co.,  
Bayonne, N. J.
- WATSON, RALPH A. .... Chief Engr., Cons. Arizona Smelt. Co., Humboldt, Ariz.
- WILDER, FRANK A. .... Pres., Southern Gypsum Co. Inc., North Holston, Va.
- WILLIAMS, GEORGE K., Chief Mech. Engr., Chile Exploration Co.,  
Chuquicamata, Chile.
- WILSON, MALCOLM EARL, Asst. State Geol., Mo. Bureau of Geology & Mines.  
Rolla, Mo.
- WORCESTER, RICHARD HOE, Supt. Min. Dept., Laclede-Christy Clay Products Co.,  
5900 Manchester Ave., St. Louis, Mo.

### Associate Members

- MERRITT, WILLIAM E. .... Supt., Tennessee Chemical Co., Nashville, Tenn.
- POPE, CHARLES E., Pres., Impervious Metal Corp.,  
421 Wood Street, Pittsburgh, Pa.
- SCHMIDT, WALTER CHARLES, Townsite Engr., Consolidated Arizona Smelting Co.,  
Humboldt, Ariz.
- THATCHER, LE ROY. .... Gen'l. Mgr., Union Mines Co., Union Mines, Nev.
- WEISS, ROBERT W., Machinery Salesman, Allis-Chalmers Mfg. Co.,  
Casilla 2653, Santiago, Chile.

### Junior Members

- ASMUS, LEWIS DURAND ..... Student, Drexel Institute, Philadelphia, Pa.
- BRYAN, JOHN RAYMOND ..... 621 W. 6th St., Reno, Nev.
- BUGBEE, JAMES MARION ..... Student, Mass. Inst. of Tech., Cambridge, Mass.
- CLARKE, ROSCOE H. .... 410 W. Tureo St., Oklahoma City, Okla.
- FRASER, THOMAS, 2d Lieut. Engrs., U. S. R., Co. 7, Engineer Officers'  
Training School, Fort Leavenworth, Kans.
- HIGGINS, ROBERT, Testing Dept., International Smelt. & Ref. Co., Tooele, Utah.
- HOLMES, LAURENCE A., Road Inspector, Board County Road Commissioners,  
Houghton, Mich.
- HUSSISSIAN, KHOREN L. .... Student, University of Wisconsin, Madison, Wis.
- HYATT, CHARLES SCOTT, Student of Met., Lehigh University,  
Alpha Tau Omega House, So. Bethlehem, Pa.
- KENDALL, ARTHUR ..... Min. Engr., Quincey Mining Co., Hancock, Mich.
- KINSLEY, ARTHUR CARRUTHERS, Student, Colorado School of Mines,  
Box 5, Golden, Colo.
- KNEPPER, CHESTER M. ... Captain, U. S. Navy, retired, 19 W. Main St., Somerset, Pa.
- LAWSON, STUART C., Asst. in Testing Dept., Anaconda Copper Mining Co.,  
Washoe Smelter, 521 Oak St., Anaconda, Mont.
- MUNN, HAROLD E. .... Engr., Utah Fuel Co., Sunnyside, Utah.

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| NOBLE, CLARKE OSBORNE.....  | 802 E. Silver Ave., Albuquerque, New Mex.            |
| PETERSON, ARTHUR F.....   | Anvil Club, Bessemer, Mich.                          |
| RALPH, THOMAS GRAHAM.....   | Student, Lehigh University, So. Bethlehem, Pa.       |
| SANDLER, LOUIS.....   | Met. Engr., Johnson Bronze Co., New Castle, Pa.      |
| SCHEURER, LEROY ROBERT.....   | Student, Missouri School of Mines, Rolla, Mo.        |
| SKEEN, LESLIE C.....  | Student, Missouri School of Mines, Rolla, Mo.        |
| TAYLOR, CARROLL CUSHING, Coast Artillery Corps, U. S. Army,                 |  |
|   | 25 Fairview Ave., Plainfield, N. J.                  |
| TERRY, MARK LOREN.....  | Student, Missouri School of Mines, Rolla, Mo.        |
| VANCE, STRATTON.....  | Student, Lehigh University, So. Bethlehem, Pa.       |
| WEIMER, RAYMOND S.....  | Student, Missouri School of Mines, Rolla, Mo.        |
| WHITNEY, LEONARD H.....   | Officers' Reserve Training Camp, Fort Sheridan, Ill. |
| ZIEGLER, FRANK K., Laboratory Asst., U. S. Bureau of Mines, Bee Ridge, Fla. |  |
| ZOLLER, LAWRENCE J.....   | Valerius, McNutt & Hughes, Tulsa, Okla.              |

**Total Membership, Aug. 10, 1917..... 6,456**

### CANDIDATES FOR MEMBERSHIP

**APPLICATION FOR MEMBERSHIP.**—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period July 10, 1917, to Aug. 10, 1917, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

#### *Members*

**William Rose Benzie, Denver, Colo.**

Proposed by Fred Carroll, P. M. McHugh, Fred H. Bostwick.

Born 1864, Stratford, Ont., Canada. 1885, Public Schools, Chicago, Ill. 1887, Night Law School. 1895–1902, Mining, Cripple Creek, Arizona, Nevada, Colorado. Engaged in mining in Colorado since 1902.

Present position: Sec'y. and Treas., Michigan Min. & Mill Co.

**Robert Sutherland Billings, Kingman, Ariz.**

Proposed by W. G. Swart, J. A. Porter, T. B. Stearns, Henry Kehoe.

Born 1858, St. George, New Brunswick, Canada. 1880, Pittsburgh and Kokomo Smelting Co., Kokomo, Colo. 1881–82, California Smelting Co., Leadville, Colo. 1883–90, Pueblo Smelting & Refining Co., Philadelphia Smelting & Refining Co., Pueblo, Colo. 1890–1900, Mining and Cons. Engr., for a number of mining companies. 1903, Buyer, Rocky Mountain Smelting & Refining Co., Florence, Colo. 1907–08, Gen'l. Mgr., Independent Smelting Co., Golden, Colo.

Present position: Mgr. and Cons. Engr., for a number of min. companies.

**Frank Oliver Cartwright, Rancagua, Chile, S. A.**

Proposed by A. E. Swartz, W. J. Turner, F. H. F. Hampton.

Born 1895, Walsall, Staffordshire, Eng. 1899–1903, Wyrley Public School, Wyrley, Staff., Eng. 1903–09, Central School, Preston, Idaho. 1909–11, Science

& Arts Inst., Walsall, Staff., Eng. 1911-13, O. S. Academy, Preston, Idaho; Min. Engrg. student, I. C. S.; Student, Alexander Hamilton Institute. 1913, Accountant; York-Ontario Silver Mines, Ltd., Cobalt, Ont., Canada. 1914-17, Accountant and statistician, The Dome Mines Co., Ltd., So. Porcupine, Ont., Canada.

Present position—1917 to date: Sec. to Smelter Supt. and Met. Bookkeeper, The Braden Copper Co.

**George W. Coffey**, Thane, Alaska.

Proposed by E. V. Daveler, John Z. Bayless, G. T. Jackson.

Born 1895, Nevada City, Cal. 1913-17, B. S., Univ. of California. 1915, summer, Brunswick Min. Co., Grass Valley, Cal. 1916, summer, Nevada Cons. Copper Co., Ely, Nev.

Present position: Mine Operator, Alaska Gastineau Min. Co..

**Aquiles Concha S.**, Santiago, Chile.

Proposed by Huntington Adams, Henry Hay, W. Lorrain Cook.

Born 1885, Santiago, Chile. 1902-05, School of Arts & Industries, Santiago; Diploma of electricity. 1906-11, sent by Governor of Chile to Paris; 1 year at Duvignau; 2 years Nationale Supérieure des Mines, 2 years Speciale des Travaux Publics. 1912-15, Bureau of Mine Inspection, Dept. of Public Works, Engineer, then mineralogist, and for the past two years, geologist.

Present position: Geologist, Bureau of Mine Inspection.

**Joseph Abraham Daniel**, Montreal, Can.

Proposed by John H. Banks, Kirby Thomas, John E. Hardman.

Born 1881, Bombay, India. 1903-06, Grad., Cambridge Univ., B. A. 1910-12, Studied at Royal School of Mines, So. Kensington, London. 1913-14, Royal School of Mines. 1915, elected Associate, Institution of Mining and Metallurgy; Member, Institution of Mining Engineers, London. 1907-09, Asst. Mineral Surveyor. 1909-10, Acting Principal, Mineral Surveyor, Government of Ceylon. 1912-13, So. Nigeria for the African Assoc. 1914-15, So. India, mining, private concern. 1915-16, In So. Amer., Chile and Bolivia for Simon Patino.

Present position—1916 to date: Min. Engr. and Geol., Hudson's Bay Co., London.

**Berd Wallace Dunn**, Spokane, Wash.

Proposed by L. K. Armstrong, Frederic Keffer, J. McD. Porter.

Born 1879, Houghton, Mich. 1896-1900, Michigan College of Mines, Houghton, Mich. 1900-01, Mgr., Belt Mine, Mich. 1901-03, Reporting Engr., Mexico. 1903-08, Reporting and operating miner, Ontario, Canada. 1908-10, Reporting Engr. 1910-11, Geological work, Porcupine and Cobalt, Ont., Canada. 1911-16, Cons. Geol., Engr., Calgary, Alberta. 1916-17, Mgr., Highland Valley Mine, Ashcroft, B. C., Canada.

Present position: Genl. Mgr. and Vice-Pres., U. S. Magnesite Co.

**Charles Pomeroy Fiske**, Palmerton, Pa.

Proposed by Francis P. Sinn, George C. Stone, J. H. Polhemus.

Born 1882, Buffalo, N. Y. 1901, Grad., Buffalo High School. 1907, Grad., Chem. Engrg., Univ. of Pennsylvania. 1907, Chemical Laboratory; 1908, Research Dept; 1910, Spiegel Dept; 1911, Research Dept; 1913, Asst. Chief, Research Dept; 1916, Chief Research Dept., New Jersey Zinc Co. (of Pa.), Palmerston, Pa.

Present position: Chief of Spelter Dept., New Jersey Zinc Co. (of Pa.).

**Jacob M. Frankel**, Morenci, Ariz.

Proposed by P. B. Scotland, John Kiddie, Frank D. Rathbun.

Born 1889, New York, N. Y. 1907, Grad., High School of Commerce, New York, N. Y. 1913, Grad., Colorado School of Mines, Golden, Colo., E. M. 1912, summer, Terico Mine, Colorado Fuel & Iron Co., miscellaneous work. 1913, Cerillos Coal & Coke Co., Madrid, N. M., miscellaneous work.

Present position—1914 to date: Private Secretary to Gen. Supt. of Mines, Arizona Copper Co., Ltd.

**Waldron Alvord Green**, Jarbidge, Nev.

Proposed by E. A. Austin, Donald Steel, Axel M. Peterson.

Born 1894, Pasadena, Cal. 1909-10, Pasadena High School. 1911-12, Palo

Alto High School. 1913-16, Stanford Univ., A. B. 1913, Underground work, Eagle Shawmut Min. Co., Tuolumne Co., Cal. 1914, Tool Dresser, Placerita Oil Co., Los Angeles Co., Cal.

Present position—1916 to date: Engr. and Geol., Elkoro Mines Co.

**John Faber Hanst, Antofagasta, Chile.**

Proposed by W. L. DuMoulin, Arthur K. Adams, L. R. Wallace.

Born 1885, Wilkes-Barre, Pa. 1900-03, Central Manual Training High School, Phila., Pa. 1903-07, Lehigh Univ., E. M. 1905, Transittman, W. G. Wilkins Engr. Co., Pittsburgh, Pa. 1906, Transittman, Morgantown & Kingwood R. R. (Construction). 1907-10, Min. Engr., Staff of Cleveland Cliffs Iron Co. 1910, Chief Engr., Cortes Assoc. Mines, Jacala, Hgo., Mex. 1910-17, Min. Engr. and Geol., Special Shaft and Construction Engr., Cleveland Cliffs Iron Co.

Present position: Chief Min. Engr., Andes Copper Min. Co.

**Marden Warner Hayward, Denver, Colo.**

Proposed by Otto Sussman, H. L. Brown, O. R. Whitaker.

Born 1882, Yankton, S. D. 1906, Mass. Inst. of Tech., S. B. in Mining. 1907, S. B. in Geology. 1907-08, Utah Copper Co., and Ray Consolidated Copper Co. 1909-10, Geol., Spurr & Cox. 1911-15, Private practice, Cons. Min. Engr. and Geol.

Present position: Engr., The American Metal Co., Ltd.

**Waldemar Frederic Henniger, Tampico, Tamps., Mex.**

Proposed by R. H. Soper, Edwin A. McKanna, C. W. Hamilton.

Born 1883, Fayetteville, Texas. 1913-16, Univ. of Texas. 1914, summer, Asst. Geol., J. M. Guffey Petroleum Co., Wichita Falls, Texas. 1915, summer, Asst. Geol., Gulf Production Co., Wichita Falls, Texas. 1916, summer, Geol., Gulf Production Co., Houston, Texas.

Present position—Nov. 1916 to date: Asst. Geol., Mexican Gulf Oil Co.

**Charles K. Hitchcock, Jr., Bonne Terre, Mo.**

Proposed by N. A. Stockett, C. J. Adami, R. S. Foster.

Born 1877, Bayonne, N. J. 1893-97, A. B. 1897-1900, E. M., Columbia School of Mines, New York, N. Y. 1900, Engr., Tenn. Coal, Iron & R. R. Co., Pratt City, Ala. 1900-01, Asst. in Min., Columbia University, New York, N. Y. 1901-07, Min. Engr., Quincy Min. Co., Hancock, Mich. 1907-10, Supt., Adventure Cons. Copper Co., Greenland, Mich. 1910-14, Supt., Lake Copper Co., Lake Mine, Mich. 1914, Min. Engr., Oliver Min. Co., Virginia, Minn.

Present position: Min. Engr., St. Joseph Lead Co.

**Bryce Hutchison, Berwick, Iowa.**

Proposed by S. W. Beyer, L. E. Young, Thomas T. Read.

Born 1884, Zenosville, Iowa. 1904, Grad., Ames High School. 1909, Grad., Iowa State College, Ames, Iowa, B. S. 1909-13, Min. Engr., Crescent, Coal Co., Oskaloosa, Iowa.

Present position: Min. Engr. and Asst. Supt., Norwood-White Coal Co.

**Edward F. Kenney, Philadelphia, Pa.**

Proposed by M. G. Moore, D. M. Stackhouse, George E. Thackray.

Born 1869, Philadelphia, Pa. 1883-88, Univ. of Pa., B. S. and C. E. 1889-1907, Asst. Engr. and Construction Engr. of Tests, Penna. R. R. Co. 1907-17, Met. Engr., Cambria Steel Co.

Present position: Met. Engr., Subsidiary Companies of Midvale Steel & Ordnance Co.

**J. H. Lewis, Rancagua, Chile, S. A.**

Proposed by Ross E. Douglass, W. J. Turner, B. T. Colley.

Born 1881, Harleek, North Wales. 1890-97, Grammar Schools, Tacoma, Wash. 1903-04, Special, Washington State College, Pullman, Wash. 1898-1900, Miner, War Eagle Mine, Rossland, B. C. 1900-02, Millman and amalgamator, Yellowstone Gold Mine, Salmo, B. C. 1906-07, Millman, Standard Mill, Wallace, Ida. 1907-09, Shift boss, Nevada Cons. Mill, McGill, Nev. 1909, Mill Supt., Glasgow Western Exploration Co., Cherry Creek, Nev. 1910-1915, Foreman, Crushing plant, and general mill repair foreman, Nevada Cons., McGill, Nev. 1916, Genl. Mill Foreman, U. S. Tungsten, Ely, Nev.

Present position: Asst. Mill Supt., Braden Copper Co.



**Jesse F. McDonald, Leadville, Colo.**

Proposed by Charles Loughridge, P. M. McHugh, Fred Carroll.

Born 1858, Ashtabula, Ohio. Public Schools and Academy. 1905, Colorado School of Mines, E. M. 1887-98, Mgr., Robinson Consolidated Mining Co. Since that time have been employed in my own mines and other affairs.

Present position—1915 to date: Mgr., Down Town Mines Co.

**Erwin Huntington MacDonald, Spokane, Wash.**

Proposed by L. K. Armstrong, J. McD. Porter, S. Shedd.

Born 1869, Morristown, N. Y. 1893, Grad., Michigan College of Mines, E. M. 1893-95, Prof. of Engineering, College of Mont., Deer Lodge, Mont. 1895-96, Foreman, Concentrating Dept., Anaconda Copper Min. Co., Anaconda, Mont. 1896-97, Draftsman, Foundry Dept., same company. 1898-1901, Civil and Min. Engr., Anaconda. 1902-06, Prof. Min. Engineering, State School of Mines, Butte, Montana. 1907-08, Min. Engr., Butte, Mont.

Present position: Min. Engr., Spokane, Wash.

**Perry North Moore, New York, N. Y.**

Proposed by C. M. Weld, C. P. Perin, S. M. Marshall.

Born 1886, Oxford, Ohio. 1904-08, Harvard Univ., A. B. 1908-10, Columbia Univ., E. M. 1910-12, Surveyor and Asst. Engr. in charge of open cut iron min. and railroad construction, Rose Run Iron Co., Olympia, Ky. 1912-16, Supt. of factory, salesman, sales manager, Euston White Lead Co.

Present position: Asst. in purchasing dept., C. P. Perin. & S. M. Marshall.

**George D. Morgan, Billings, Mont.**

Proposed by E. Carpenter, L. C. Snider, Irving Perrine.

Born 1894, Shawnee, Okla. 1917, Univ. of Oklahoma, A. B. 1913-15, Oklahoma Geol. Survey.

Present position—1916 to date: Geol. in charge, Empire Gas and Fuel Co.

**Alexander Colin Munro, Hurley, New Mex.**

Proposed by Ira L. Wright, Fidel Martinez, W. H. Elbelt.

Born 1883, Baulardarie, Cape Breton, Canada. 1889-97, Public Schools. 1897-98, North Sydney Academy, Cape Breton. 1898-99, Teaching school, Kempt Head, Cape Breton. 1914, Courses in Chemistry, Physics, Classics, Ferris School, Big Rapids, Mich. 1899-1900, Repairman, Boston & Montana Co., Great Falls, Mont. 1900-02, Repairman and Foreman, Anaconda Copper Mining Co., Anaconda, Mont. 1904-11, Shift Boss and Gen'l Repair Foreman, Cananea Cons. Copper Co., Sonora, Mex. 1911-12, Gen'l Repair Foreman, Ray Cons. Copper Co., Ray, Ariz. 1912-14, Gen'l Mill Foreman, Braden Copper Co., Chile.

Present position—1914 to date: Gen'l Foreman, Tailings Disposal and Retreatment Plant.

**Jackson A. Pearce, Idaho Springs, Colo.**

Proposed by P. M. McHugh, Charles A. Chase, Fred Carroll.

Born 1876, San Bernardino, Cal. 1892, Grade School; 1892-96, High School, San Bernardino, Cal. 1896-1901, and 1902-06, Univ. of Cal., B. S., & M. S. 1897-1901, Asst. in Chem., Univ. of Cal. 1901-02, Agricultural Chem., Spreckels Sugar Co., Watsonville, Cal. 1902-06, Instructor in Chem. and other sciences, Berkeley High School. 1906-07, Min. and Mill., Delta Consolidated Mines Co., Cal. 1907-10, Assaying, min. and mill. in Nev. for Mispah Min. Co., Nevada Hills Min. Co., Montana Tonopah Min. Co. 1910-12, Mgr., Reduction Works, Trench-Bobs Mines Co., So. Africa. 1912-13, Head Refiner, Dome Mines Co., Can.

Present position—1913 to date: Supt., Argo Mill, Idaho Springs, Colo.

**Herbert E. Robinson, Jarbidge, Nev.**

Proposed by E. A. Austin, Donald Steel, Axel M. Peterson.

Born 1883, Sloan, Iowa. 1908, Iowa State College, B. S.; 1914, E. M. 1903-04, Horseshoe Min. Co. 1905-08, Homestake Min. Co. 1908-09, Ruby Gulch Min. Co., 1909-10, Minerva Min. & Mill. Co. 1910-15, North Star, Bluster Alpha Min. Co.

Present position—1916 to date: Engr., Elkoro Min. Co.

**Percy Hoke Royster, Pittsburgh, Pa.**

Proposed by Alexander L. Feild, Arthur E. Wells, Dorsey A. Lyon.

Born 1888, Raleigh, N. C. 1907, Univ. of North Carolina, A. B.; A. M., 1908.

1908-15, Harvard Univ., A. M. 1915-16, Turbine Research Dept., General Electric Co., Lynn, Mass. 1916, Research Engr., Cutting & Washington, Cambridge, Mass., manufacturers of radio apparatus. Member Institute of Radio Engineers.  
Present position—1916 to date: Junior Physicist, U. S. Bureau of Mines.

**John Shanks, Nordegg, Alberta, Can.**

Proposed by Lewis Stockett, B. L. Thorne, W. F. McNeill.

Born 1873, Kelsyth, Scotland. 1886, Board School, Kelsyth, Scotland. 1886-92, Allen Glen Tech. College, Glasgow. Certificates in advanced stages of mathematics, electricity, organic and inorganic chemistry, theoretical and applied mechanics, machine construction, Queen's Medalist, South Kensington, London. 1892-95, Mech. Engr., and Draftsman, William Baird & Co., Ltd., Bothwell & Kelsyth, Scotland. 1895-98, Apprentice Engr., Simpson & Wilson, Glasgow. 1898-1901, Min. Engr., Whitehaven Colliery Co., Ltd., Whitehaven, Cumberland, England. 1901-02, Mine Mgr., James Bain & Co., Harrington Collieries, Cumberland, England; 1902-08, Asst. to mining partner, J. Bain & Co.; Cons. Engr., Whitehaven Collieries. 1910-14, Supt., Coal Creek Mines, Crowsnest Pass Coal Co., Ltd., Fernie, B. C. Can.  
Present position: Gen'l Mgr., Brazeau Collieries, Ltd.

**John Elmer Thomas, Jr., Tulsa, Okla.**

Proposed by F. Julius Fohs, L. L. Hutchison, W. E. McCourt.

Born 1892, Toledo, Ohio. 1905-09, Toledo Central High School. 1909-12, Univ. of Chicago. 1912-13, Asst. to Chief Geol., American Smelting & Refining Co., Sierra Mojada, Coah., Mex. 1914 (July), E. N. Gillespie, Tulsa, Okla.; (Aug) Wichita Natural Gas Co., Bartlesville, Okla. 1914-15, Geol., Minnehoma Oil Co., Tulsa, Okla. 1915-16, Chief Geol., Chicago Oil Co., Chicago, Ill. 1916-17, Chief Geol., Sinclair Oil & Gas Co., Tulsa, Okla.  
Present position: Cons. Geol.

**Nelson C. Titus, Spokane, Wash.**

Proposed by L. K. Armstrong, J. McD. Porter, J. V. Richards.

Born 1880, Carmi, Ill. 1896, Millman, Penobscot mine; Northern Mines Co., Ltd., Marysville, Mont. 1901, Amalgamator, Silver Star Mill, Silver Star, Mont. Mill Construction, Gold Gulch Min. Co., Mont. 1902, Met. Investigating King Solomon Mine, Gazelle, Cal. 1903, Met. and Design of plant, California Mines, Ltd., Sumpter District, Ore. 1905, Cleveland Mines, Ida., Pullman Mine & Mill, Gamble Gulch, Colo. 1906, Construction of 300 ton cyanide plant, Waltham Min. & Mill Co., Idaho Springs, Colo. 1907, Hermosa Copper Co. Erected 100 ton concentrator. Tests in pyritic smelting and operating furnaces, Comanche Smelter, Silver City, N. M. Associated in firm of Bryant & Titus, Silver City, N. M. 1909, Mgr., Aurora Min. Co., Honduras. U. S. Mineral Surveyor, Susitna River. Engr. and Salesman, Fairbanks Morse & Co., Seattle. 1914, Consultant, Hall Creek Min. & Mill Co., Ferry Co., Wash. Examination, Eureka Mining District, Nev.; Shaughnessy Hill Property, Libby, Mont. Report Summit Gold Prospects near Loomis, Wash.; Okanogan Gold Mines. 1915, Examinations, Coeur d'Alene District. Report on property of Western S-L Co., near Sweeney, Ida. 1917, Examination, Jarbidge, Nev.  
Present position: Managing Director, Orient Metals Min. & Smelt. Co. and subsidiaries.

**Ray E. Tower, Marysville, Mont.**

Proposed by Wm. R. Price, J. H. McCormick, Rudolph Emmel.

Born 1889, Missoula, Mont. 1896-1905, Grammar and High Schools, Butte, Mont. 1906, Miner and sampler, Colusa-Parrat Min. & Smelt. Co. 1907-09, Assayer and chem., Easton-Pacific Min. Co. 1910-11, Mill Supt., Barton Development Co. 1912-13, Chem., Anaconda Copper Co. 1914, Independent operator.  
Present position: Mgr., Tower & Templeman Leasing Co.; Supt., St. Louis Min. & Mill Co.

**Charles Sydney Trewin, New York, N. Y.**

Proposed by J. A. Singmaster, Wm. H. Shearman, Thomas T. Read.

Born 1891, Phila., Pa. 1909, Stevens Prep., Hoboken, N. J. 1913, Stevens Tech., Hoboken, N. J., M. E. 1913-15, Power Division. 1915-17, Technical Asst. to Genl. Supt., New Jersey Zinc Co., (of Pa.).  
Present position: Mech. Engr., Tech. Dept., New Jersey Zinc Co.

**William George Waring**, Webb City, Mo.

Proposed by Geo. C. Stone, Matthew van Sicken, W. B. Shackelford.

Born 1847, Cedar Creek, Pa. 1861-64, State College of Penn., M. S. 1867-79, Mine surveying; horticultural pursuits. 1879-99, Asst., U. S. Geological Survey, Denver; also Met. for several min. companies. 1899-1917, Assay Office and Research Laboratories, Joplin and Webb City, Mo. 1882-85, Asst. Supt., Rico Reduction Wks., Rico, Colo.; Mgr., Hope & Cross Mine. 1886-89, Supt. and Mgr., Flagler Reduction Wks., Silver City, N. M. 1889-94, Met. and Cons. Engr., Hidalgo Min. Co., Parral, Chih., Mex. 1894-99, Cons. Engr., Moctezuma Min. Co., Sonora, and Hidalgo Min. Co., Parral. 1899-1917, Prop., Waring Laboratories, Webb City, Mo. Present position: Cons. Chem., Waring & Williams Laboratories.

**Norman L. Wimmeler**, Shanghai, China.

Proposed by H. Foster Bain, John Wellington Finch, Milnor Roberts.

Born 1885, Alma, Wis. Minneapolis Central High Schools. 1908, Grad., School of Mines, Univ. of Washington. 1909, Supt., Nicola Dev. Co., Merritt, B. C., Special coal examination, U. S. G. S. 1909-10, Examination for private interests. 1911, Min. Engr., Kenai Alaska Gold Co., Seward, Alaska. 1912, Min. Engr., Gold Bullion Min. Co., Knik, Alaska. 1913-14, Cyanide and Refining, Alaska Treadwell Co. 1914-16, Supt., dev. work, Alaska and British Columbia, Herman Innes Co. 1916, Exploration in Philippine Islands, Herman Innes Co.

Present position: Min. Engr., New York Syndicate, Orient.

**Roy U. Wood**, Garfield, Utah.

Proposed by R. W. Senger, R. F. Barker, A. H. Richards.

Born 1890, New York, N. Y. 1905-09, DeWitt Clinton High School, New York, N. Y. 1909-10, Columbia College. 1910-14, School of Mines, Columbia Univ., Met. Eng. 1914, summer, Instructor, Columbia Univ. summer school of surveying, Morris, Conn. 1914-15, Asst., Dept. of Met., Columbia Univ. 1915, Chem., American Smelting & Refining Co., Perth Amboy, N. J. 1915-17, Chem., Garfield Smelting Co., Garfield, Utah.

Present position: Asst. to Met., Garfield Smelter.

**William Ryer Wright**, New York, N. Y.

Proposed by Clark G. Mitchell, Robt. Peele, Wm. Campbell.

Born 1888, New York, N. Y. 1907, Stevens Prep., Hoboken, N. J. 1907-11, School of Mines, Columbia Univ., New York, N. Y., E. M. 1907, Hudson Companies, Ltd., Levelman, Pier C. Shaft, Jersey City, N. J., McAdoo Tunnel. 1911-12, Asst. Mine Engr., Braden Copper Co., Rancagua, Chile. 1913-14, Member, Stafford & Wright, Engineers and Contractors, New York, N. Y. 1914-15, Engr., W. & W. F. Crockett, New York, N. Y. 1915-16, Engr., Construction and Chemical, Aetna Explosives Co., Inc., New York, N. Y. 1916-17, Sales Engr., Denver Rock Drill Mfg. Co., Denver, Colo.

Present position: Asst. Engr., Aetna Explosives Co., New York, N. Y.

**Chuji Yabe**, Ehimeken, Japan.

Proposed by Wataru Watanabe, Benzo Katsura, Tadashi Inouye.

Born 1886, Tokushima, Japan. 1892-1905, Common School. 1905-08, 3d High School, Kyoto. 1908-11, Kyoto Imperial Univ., degree of Koyakushi. 1911, Min. Engr., Sumitomo Besshi Copper Mine.

Present position: Min. Engr., Sumitomo Besshi Copper Mine.

### *Associate Member*

**Jay G. Puterbaugh**, McAlester, Okla.

Proposed by Geo. M. Brown, J. J. Rutledge, J. C. Reid.

Born 1876, Newton, Kans. 1893, Grad., Common and High Schools, Hutchinson, Kans. 1893-95, Retail coal business, Hutchinson, Kans., Carey Puterbaugh Coal Co. 1895-97, Traveling salesman, Kans. 1897-98, Dist. Sales Mgr., Kansas City. 1898-1902, Southern Sales Mgr., Dallas, Tex., Western Coal & Min. Co. 1902-03, Vice-Pres. 1903-07, Genl. Mgr., Cons. McAlester Fuel Co., McAlester, Okla. 1907-17, Pres. and principal owner, McAlester Fuel Co., McAlester, Okla. 1913-17, Pres., Southern Anthracite Coal Min. Co., Rosselville, Ark. 1914-17, Vice-Pres., American National Bank, McAlester, Okla. 1914-17, Chairman, Board of Directors, Dardanelle and Rosselville Railroad Co., Dardanelle, Ark.

Present position: Pres., McAlester Fuel Co., McAlester, Okla.; Pres., Southern Anthracite Coal Min. Co., Rosselville, Ark.

### *Junior Members*

**Thomas Coburn Brown**, Socorro, N. M.

Proposed by Richard S. McCaffery, W. J. Mead, Cony T. Brown.

Born 1895, Socorro, N. M. 1916, Asst. Engr., Empire Zinc Co., Kingston, N. M. 1917, Asst. Engr. on inspection trips through Chile, Bolivia, and Peru for New Jersey Zinc Co.

Present position: Student, University of Wisconsin, Madison, Wis.

**John Ripley Poss**, Houghton, Mich.

Proposed by F. W. McNair, F. W. Sperr, W. E. Hopper.

Born 1895, Caseville, Mich. 1913, Detroit University School, Detroit, Mich. 1914, Colorado School of Mines, Golden, Colo. 1915, Mich. College of Mines. 1911, summer, Abbott Motor Co., Detroit, Mich. 1912, summer, Poss Motor Co., Detroit, Mich.

Present position: Student, Mich. College of Mines.

**K. T. Sparks**, Butte, Mont.

Proposed by C. H. Clapp, C. H. Bowman, Theodore Simons.

Born 1893, Pocatello, Ida. 1908-10, Butte High School. 1910-13, Butte Business College. 1914-17, Montana State School of Mines, E. M. 1910-17, summers, Miner, Anaconda Copper Mining Co., Butte, Mont.

Present position: Student, Montana State School of Mines.

### *Change of Status—Junior Member to Member*

**Thomas E. Cassilly**, St. Paul, Minn.

Proposed by A. L. Gholz, John E. Hodge, H. M. Roberts.

Born 1893, Des Moines, Iowa. 1906, Public School, Des Moines, Iowa. 1913-17, Grad., University of Minnesota, Minneapolis, Minn., E. M. Previous to college course worked at various mines in western camps as a mucker and miner. 1915, Contract miner, Shenago Furnace Co. 1916, Trackman, Old Dominion Mine.

Present position: Mining Engr., E. J. Longyear Co., Minneapolis, Minn.

**Michiya Hiraoka**, Osaka, Japan.

Born 1877, Japan. Grad., Imperial Tokyo University, Japan. Engr., Furukawa & Co., Ashio Copper Mines, Japan.

Present position: Prof. min. engrg. and met., Osaka Technical College.

### CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period July 10, 1917, to Aug. 10, 1917.

This list together with the list published in Bulletin Nos. 121 to 128, January to August, 1917, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1917, and brings it up to the date of Aug. 10, 1917.

|   |  |
|---|--|
| ACOSTA, J. N., Cons. Engr.                | Hotel Brewster, San Diego, Cal.                      |
| ADAMS, HUNTINGTON                         | White, Weld & Co., 14 Wall St., New York, N. Y.      |
| ALLEN, ARTHUR P.                          | Kellogg, Ida.  |
| ALLEN, C. A.                              | 329 No. Ewing Ave., Dallas, Tex.                     |
| ALLEN, HEMAN H., Private, D. Company,     | 8th Reserve Engineers,                               |
|   | American Lake, Wash.                                 |
| ANDERSON, JOHN C.                         | Arizona Cons. Copper & Min. Co., Greaterville, Ariz. |
| ANDREWS, CLARENCE W., Special Apprentice, | American Steel Foundries, Alliance, O.               |
| ASH, SIMON H.                             | 1223 Alaska Bldg., Seattle, Wash.                    |

- BACON, MAURICE W. .... 726 Old National Bank Bldg., Spokane, Wash.  
 BALL, MAX W. .... Roxana Petroleum Co., Box 930, Cheyenne, Wyo.  
 BATCHELLOR, STILLMAN. .... Cruces de Avalos, Via Guane, Pinar del Rio, Cuba.  
 BECK, EDWIN L. .... 1127 El Centro Ave., Oakland, Cal.  
 BETTERTON, JESSE O. .... 2026 Fowler Ave., Omaha, Neb.  
 BJORGE, GUY N. .... 1006 Hobart Bldg., San Francisco, Cal.  
 BLOGG, CECIL F. .... 708 Belmont Place, Seattle, Wash.  
 BONILLAS, Y. S. .... American Metal Co., Casilla 125 D, Santiago, Chile.  
 BONILLAS, YGNACIO. .... Mexican Embassy, Washington, D. C.  
 BORNHOFF, HENRY. .... Apartado 251, Monterrey, N. L., Mex.  
 BRINDLE, ARTHUR St. C. .... 1565 Wilnot Pl., Oak Bay, Victoria, B. C., Can.  
 BROWN, R. GILMAN. .... 7 Gracechurch St., London, E. C., Eng.  
 BRUNEL, LOUIS J. .... 1032 62d St., Oakland, Cal.  
 BURROUGHS, A. H., JR. .... Ruby, Mont.  
 CARNAHAN, CHARLES T. .... 2 Rector St., New York, N. Y.  
 CARNEY, FRANK D. .... Bethlehem Steel Co., So. Bethlehem, Pa.  
 CARPENTER, ALVIN B. .... 530 Citizens National Bank Bldg., Los Angeles, Cal.  
 CARPENTER, JAY A. .... Care E. E. Carpenter, Wonder, Nev.  
 CHASE, J. L. .... Geol. Dept., Gypsy Oil Co. of Tulsa, Box B., Tulsa, Okla.  
 CHATIN, AUGUST H. .... Tooele, Utah.  
 CHURCH, JOHN L. .... Montana Club, Helena, Mont.  
 CLAGHORN, CLARENCE R. .... Vintondale, Cambria Co., Pa.  
 CLAGHORN, D. MONTGOMERY. .... Supt., Durham Colliery Co., Durham, Wash.  
 COGHILL, WILLIAM H. .... Met., U. S. Bureau of Mines, Seattle, Wash.  
 COHEN, LOUIS. .... Box 43, Georgetown, Colo.  
 COLBURN, C. LORIMER. .... 530 Equitable Bldg., Denver, Colo.  
 COOK, W. L. .... Instructions to hold everything.  
 CORNELL, RUSSELL T. .... St. Joseph Lead Co., 71 Broadway, New York, N. Y.  
 COULDREY, PAUL S. .... Sound Beach, Conn.  
 CRAGIN, RODNEY S. .... Bisbee, Ariz.  
 CREGAN, JOHN F. .... The American Zinc & Chemical Co., Langeloth, Pa.  
 DAMAN, A. C. .... 2215 East 25th St., Denver, Colo.  
 DAVIDSON, LYNDALL P., Research Chemist, Electrolytic Zinc Plant,  
 Anaconda Copper Mining Co., Great Falls, Mont.  
 DOWNS, FLETCHER G. .... 367 Bergen Ave., Jersey City, N. J.  
 EASLEY, GEORGE A. .... La Nevada Tungsten Mine, Casilla 27 A, La Paz, Bolivia.  
 ELLIOTT, ARTHUR H. .... 39 Cortlandt St., New York, N. Y.  
 EVANS, G. S., Eastern Smelting Co., Ltd., Dato Kramat Works, Penang, F. M. S.  
 FABIAN, FRANCIS G. .... 1025 Peoples Gas Bldg., Chicago, Ill.  
 FARRAND, I. W. .... Raynesford, Mont.  
 FIDLAR, J. B. .... 1221 Arlington Ave., Davenport, Iowa.  
 FIELD, R. A. .... Instructed to hold everything.  
 FILTEAU, CHARLES A. .... National Mines, Ltd., Cobalt, Ont., Can.  
 FITCH, WALTER, JR., Pres. & Mgr., Walter Fitch, Jr., Inc., Mine & Tunnel  
 Contractors, Eureka, Utah.  
 FLYNN, F. N. .... Consolidated Min. & Smelt. Co. of Canada, Trail, B. C., Can.  
 GAEBELEIN, PAUL W. .... Alderson, Baker & Gaebelin, 32 Falk Bldg., Boise, Ida.  
 GAENSLEN, GEORGE R. .... 403 St. Anthony Ave., San Antonio, Tex.  
 GALLOWAY, A. D. R. .... 215 South Taylor St., South Bend, Ind.  
 GARRETT, S. G. .... Box 512, Casper, Wyo.  
 GATES, ARTHUR O. .... Instructed to hold everything.  
 GEPP, HERBERT W., Electrolytic Zinc Co. of Australasia, Risdon Road, Hobart,  
 Tasmania.  
 GILL, A. H. .... American Zinc & Chemical Co., Langeloth, Pa.  
 GOETTER, FRED B. .... 4818 Moneta Ave., Los Angeles, Cal.  
 GOODRICH, H. B. .... Dundee Petroleum Co., Tulsa, Okla.  
 GOODRICH, HARRY C., Chief Engr., Utah Copper Co., McCormick Bldg.,  
 Salt Lake City, Utah.  
 GRANT, ULYSSES S., IV. .... 1618 Ash St., San Diego, Cal.  
 GREENSFELDER, NELSON S., Stratton-Cripple Creek Min. & Dev. Co., Box 68,  
 Cripple Creek, Colo.  
 GRIER, CHARLES DENHAM. .... 4542 Brooklyn Ave., Seattle, Wash.  
 HAAS, HERBERT. .... 1428 Lafayette St., Alameda, Cal.  
 HAMILTON, WALTER R., Pomeroy & Hamilton, Cons. Min. & Pet. Engrs.,  
 Hobart Bldg., San Francisco, Cal.

- HANSON, HENRY.....424 Hill Lane, Oakland, Cal.  
HARRINGTON, HARRY G.....271 Meeker Ave., Newark, N. J.  
HART, RAY W.....United Verde Extension Mining Co., Jerome, Ariz.  
HAY, HENRY.....Box 93, Route 1, Santa Ana, Cal.  
HAYDEN, WALLACE H.....101 North St., Batavia, N. Y.  
HAYES, F. H., Headquarters First Infantry, Arizona National Guard, Naco, Ariz.  
HENDERSON, GEORGE M.....Bland, New Mex.  
HERRES, OTTO JR., Supt., Hiawatha Mine, United States Fuel Co.,  
Hiawatha, Utah.  
HIRAOKA, MICHITA, Prof. of Min. Engr. & Met., Osaka Higher Technical College,  
Osaka, Japan.  
HODGE, ARTHUR CAIRN.....Bellevue Terrace, St. Austell, Cornwall, England.  
HOFFMAN, J. F.....Empire Steel & Iron Co., Mt. Hope, via Wharton, N. J.  
HOLLISTER, SCOVILL E.....Resident Mgr., The Leviathan Mines Co., Yucca, Ariz.  
HOLMBERG, CLYDE L.....Duluth, Minn.  
HOOKER, WALTER, 75 Parliament Hill Mansions, Gospel Oak, Highgate, N.,  
London, Eng.  
HOOVER, THEODORE J.....450 Melville Ave., Palo Alto, Cal.  
HOWAT, ANDREW M.....Mokelumne Mines Co., Mokelumne Hill, Cal.  
HUDSON, ALBERT W.....Burro Mountain Copper Co., Tyrone, New Mex.  
HUGHES, WILSON W.....Care Lloyds Bank, Fowey, Cornwall, Eng.  
JOHNSTON, R. BARKER.....440 Estrada de Sacavem, Lisbon, Portugal.  
KERR, D. M., Electrolytic Zinc Co. of Australasia, Collins House,  
360 Collins St., Melbourne, Australia.  
KLUGE, HARRY A.....Federal Lead Co., Flat River, Mo.  
KOBBE, WILLIAM H.....Dallas, Tex.  
KOJIMA, JINTARO.....Furukawa & Co., Marunouchi, Tokyo, Japan.  
KRAMER, JOHN F.....109 Carsonia Ave., Mt. Penn, Berks Co., Pa.  
KUPFERSTEIN, JOSEPH T.....Box 96, Shreveport, Louisiana.  
LIDNER, P. G.....Porlamar, Margarita, Venezuela.  
LONERGAN, JAY.....1624 Washington Ave., Colorado Springs, Colo.  
LONGYEAR, R. D.....710 Security Bldg., Minneapolis, Minn.  
MCAULIFFE, EUGENE, Pres. Union Colliery Co.,  
Union Electric Light & Power Co. Bldg., St. Louis, Mo.  
MCCOY, A. W.....Drawer "S," Bartlesville, Okla.  
MCDONALD, CARLTON K.....1890 East 89th St., Cleveland, O.  
MC HARDY, ROY H.....Iron Cap Copper Co., Copper Hill, Ariz.  
MACKAY, ANGUS.....Box 357, Chewelah, Wash.  
MACNUTT, C. H., Lieutenant, #2 Tunnelling Co., Canadian Engineers, B.E.F.,  
Army P. O., London, England.  
MAGEE, JAMES F., Private, 1st Regiment of Engineers,  
U. S. Expeditionary Force in France, Care Adjutant General, Washington, D. C.  
MANION, EDWARD.....Guernsey, Wyo.  
MARSHALL, GEORGE B., Costa Rica Union Mining Co., Mina La Union,  
Miramar, Costa Rica.  
MATHER, T. W., Guggenheim Bros., Room 3533, 120 Broadway,  
New York, N. Y.  
MEAD, RICHARD.....Battery C, 1st Mass., F. A., East Boxford, Mass.  
MEIER, A. J.....1900 Morgan Street, St. Louis, Mo.  
MESSLER, EUGENE L.....Pres., Eureka Fire Brick Works, Mt. Braddock, Pa.  
MILLARD, H. ALFRED.....2 Rector Street, New York, N. Y.  
MILLARD, WILLIAM J., Capt. 12th Engineers (Ry.), Headquarters,  
U. S. Expeditionary Force, France.  
MONTGOMERY, HENRY S.....Box 468, Ajo, Ariz.  
MOORE, REDICK R.....Garfield Smelter, Garfield, Utah.  
MORGAN, G. H.....1010 Bryson St., Youngstown, Ohio.  
MOSIER, MCHENRY.....Box 596, Bisbee, Ariz.  
NORSWORTHY, H. R.....1249 Jones Street, San Francisco, Cal.  
NORTON, PAUL T., JR., Div. Engr., United Coal Corp., Boswell, Somerset Co., Pa.  
ORFORD, ERNEST V.....1304 Harrison Boulevard, Boise, Ida.  
PARKER, JOSEPH E.....Box 180, Bulawayo, Rhodesia, So. Africa.  
PARTANEN, ISAK.....Liberty Bell Gold Min. Co., Box 548, Telluride, Colo.  
PATES, JAMES S.....Pres. & Gen'l Mgr., Export Coal Co., Trafford, Pa.  
PATTERSON, C. T.....Met., Standard Chemical Co., Houston, Pa.  
PETTY, MORRIS K.....Mine Supt., Morris Coal Co., Cambridge, Ohio.

|   |  |
|---|--|
| POMEROY, WILLIAM A., Min. Engr., Pomeroy & Hamilton-Cons. Min. & Pet. Engrs., Hobart Bldg., San Francisco, Cal. |  |
| PRICKETT, WILLIAM C.  | 219 Ridgely Apartment, Birmingham, Ala.                    |
| PRYER, W. C.  | Alpine Apts., Anaconda, Mont.                              |
| PUTSCH, AUGUST.   | 45 Broadway, New York, N. Y.                               |
| RABLING, HAROLD,  | 69 Primrose St., Monee Ponds, Victoria, Australia.         |
| REHFUSS, WILLIAM CLIFFORD,  | 1417 South Broad St., Philadelphia, Pa.                    |
| REIFSPNEIDER, LE B. B.  | Asst. Supt., Mina Candida, Pinar del Rio, Cuba.            |
| REQUA, MARK L.  | 2306 Massachusetts Ave., Washington, D. C.                 |
| RETTET, HERMAN H.   | Moctezuma Copper Co., Nacozari, Sonora, Mex.               |
| RICE, JOHN A.   | 2336 Vine St., Berkeley, Cal.                              |
| RIDDELL, GUY C.   | Instructed to hold everything.                             |
| RISTEDT, E. J.  | 7121 Broadway, Cleveland, O.                               |
| ROBERTS, T. C.  | Sea Gull Apts., Long Beach, Cal.                           |
| ROBERTSON, J. D.  | Box 782, Carlinville, Ill.                                 |
| RUTHERFORD, FOREST.   | First National Bank, Pueblo, Colo.                         |
| SCHMIDT, HERBERT E.   | 62 Penobscot Loc., Hibbing, Minn.                          |
| SCHNEIDER, ALBERT F.  | 604 West 7th St., Plainfield, N. J.                        |
| SCHOLZ, CARL, Cons. Min. Engr., Chicago Burlington & Quincy Railroad Co., Chicago, Ill.                         |  |
| SCHUYLER, WALTER S.   | Alaska Mine, Pike, Cal.                                    |
| SEMPLER, R. A., Sapper, Canadian Engineers Training Dept., St. Johns, Quebec, Canada.                           |  |
| SEVERY, CLARENCE L., Pemberton & Severy, Cons. Geologists, 216 Unity Bldg., Tulsa, Okla.                        |  |
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| Date of Election. | Name.              | Date of Death. |
|-------------------|--------------------|----------------|
| 1885              | Bell, Robert.      | June 18, 1917. |
| 1894              | Rodgers, M. K.     | June —, 1917.  |
| 1878              | Thies, Adolf       | July 30, 1917. |
| 1915              | Winslow, Sidney W. | June 18, 1917. |



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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Oxide of Zinc

BY GEORGE C. STONE,\* PH. B., NEW YORK, N. Y.

(St. Louis Meeting, October, 1917)

THE method of making oxide of zinc direct from the ore was invented and developed at the works of The New Jersey Zinc Co. at Newark in the middle of the last century. The process was invented by Burrows, who had not the ability, financial or technical, to work out the details necessary to make it of commercial value. This was done by Col. Wetherill, whose name is commonly attached to this process. The grate bars used are also frequently called Wetherill grates, which is a misnomer because they were in common use for boiler firing at the time and he never claimed their invention.

The invention of the process was due to the efforts of The New Jersey Zinc Co. to find a profitable means of working the ores from Franklin and Sterling Hill, N. J. These are a mixture of franklinite, willemite and zincite, containing about 20 per cent. of zinc. The first attempts were to make spelter, which were not successful, owing to the low grade of the ore and the fusibility of the residue. Failing in this, the next attempt was to make oxide in large muffles and reverberatory furnaces. This succeeded, although the cost of operation was high, the recovery low, and the quality of the product uncertain. In 1855, the new process was patented and has been in successful operation ever since.

Essentially, the process is to spread a mixture of coal and ore on a body of burning coal on a perforated grate and blow an excess of air through the grate. The zinc is reduced in contact with the coal, volatilized and burned by the excess of air in the upper part of the furnace and in the flues. It is then carried to the bag rooms by the excess air and products of combustion which are forced through the flues by fans. In its main features, the process is the same today as at the time of its invention, but the details have been so modified that it would hardly be recognized by its originators.

## ORES

The process is applicable to all the oxidized ores of zinc and to roasted sulphides, provided the gangue is not so fusible as to leave a residue that is impervious to the blast. In many cases the ores contain impurities that make it impossible to produce an oxide of a good enough color to be

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\*Chief Metallurgist, New Jersey Zinc Co.

available as a pigment. These impurities are any volatile metals that form colored oxides or sulphides. Cadmium is one of the worst, because it is very volatile and its dark brown oxide and bright yellow sulphide both have strong tinctorial powers, so that small fractions of a per cent. seriously injure the color of the oxide. Lead, which is one of the commonest impurities in zinc ores, also injures the color, though to a much less degree, as it usually forms basic sulphates which are nearly white. On other accounts lead is often objectionable, particularly when the oxide is to be used for the manufacture of rubber goods. Sulphur may occur in oxide as sulphides, sulphates or as sulphurous anhydride, the first and last of which are objectionable; the first on account of probable injury to the color while the last is believed by many paint manufacturers to have a bad effect on the grinding properties. Opinion on the latter point is by no means unanimous. Sulphates, if soluble, have very objectionable qualities for outside paints as they leach out on the paint coat, leaving discolored spots. Chlorides, in appreciable quantity, are rare, but if present would be open to the same objection.

#### FUEL

The fuel used must be one that does not give a black smoke which would ruin the color of the oxide. Anthracite is the most satisfactory fuel for the purpose and the one generally used. In the West, where anthracite is expensive, semi-bituminous coals, low in volatile qualities, are sometimes substituted for it in whole or in part but they are never as good. Coke alone does not work well, as it causes a very intense local heat that makes the residue impervious to the blast. It is occasionally used as a part of the charge fuel.

#### FLUXES

With siliceous ores, limestone is sometimes added, but in general it does more harm than good, owing to its tendency to make the residue too fusible.

#### PREPARATION OF CHARGE

The ores and coal are usually received at the works in fine enough condition for use without further crushing. Nearly all the ores are either concentrates or roasted products and are inevitably finer than is necessary. As the small sizes of coal are suitable for the purpose, and are cheaper than the large, they are invariably used. If fluxes are used they are crushed to about the size of the ore and coal. There should not be a great difference in the size of the different materials, or it will be impossible to mix them properly.

## PALMERTON PLANTS

The Palmerton plants of The New Jersey Zinc Co. (of Pennsylvania), are the largest and best equipped oxide plants in the world.

Fig. 1 shows the general arrangement of the West plant and Fig. 2, that of the East. They are similar, but the East plant has fewer and larger bag rooms and differs somewhat in details. At both plants the raw materials are delivered on the trestle at the south of the plant and are either placed in bins or stocked in piles. The ore and coal are taken from the bins to the mix-house by larries. The mixed charges are delivered to the bins above the furnaces and distributed by traveling cranes.

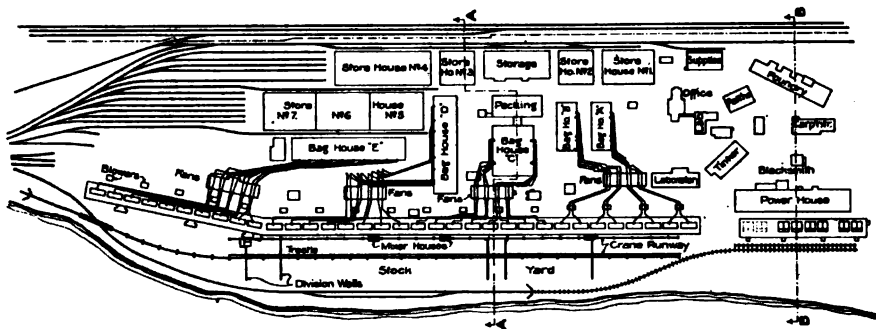


FIG. 1.—PLAN OF WEST PLANT.

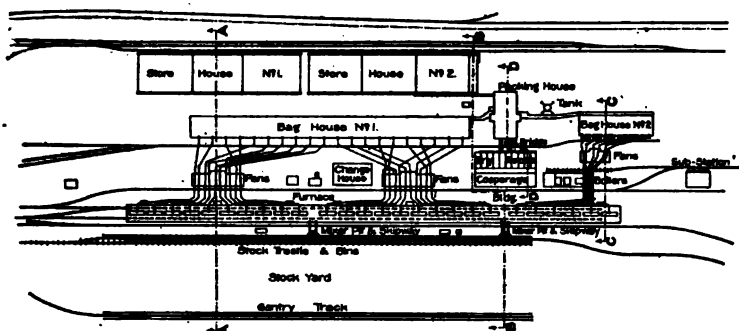


FIG. 2.—PLAN OF EAST PLANT.

Each block of furnaces is a separate unit, having its own blower, exhaust fan and bag room. The oxide is trucked from the bag rooms to the packing room and the final packages skidded or trucked to the storehouses.

*Stocking and Mixing—East Plant*

The ore and coal are delivered on the double track, steel and concrete trestle, 22 ft. (6.7 m.) high with Brown tangential bins below the track next the furnace room. Materials for current use are dumped

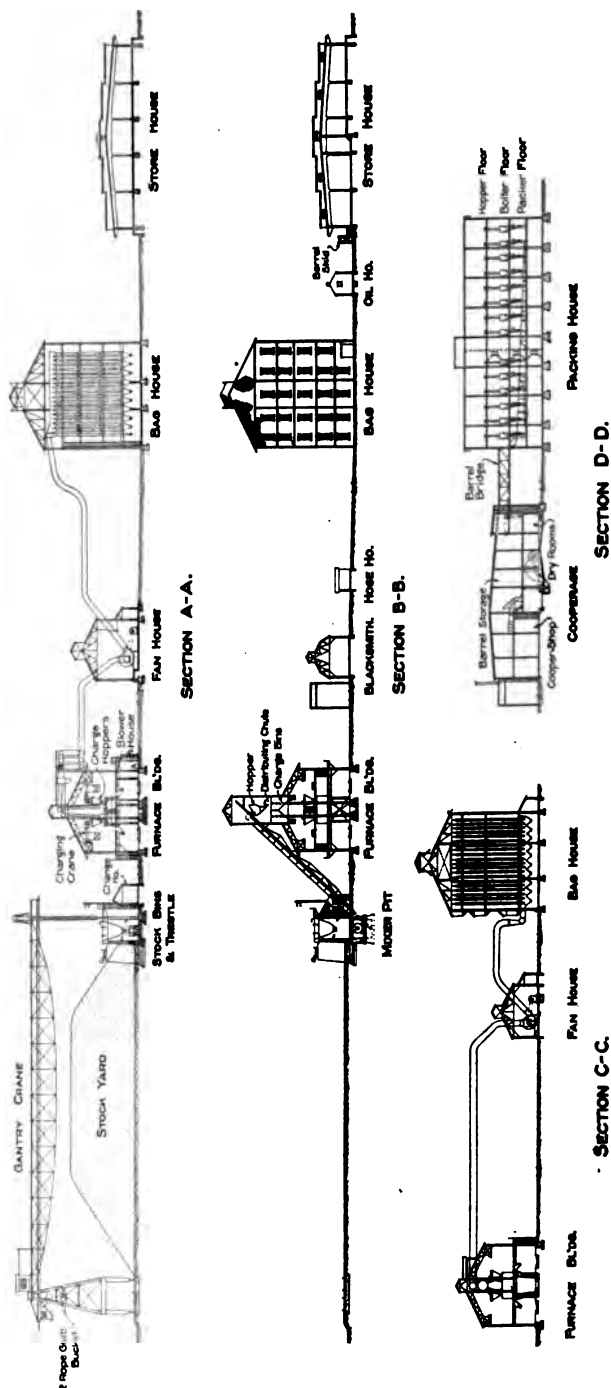


FIG. 3.—SECTIONS OF EAST PLANT.

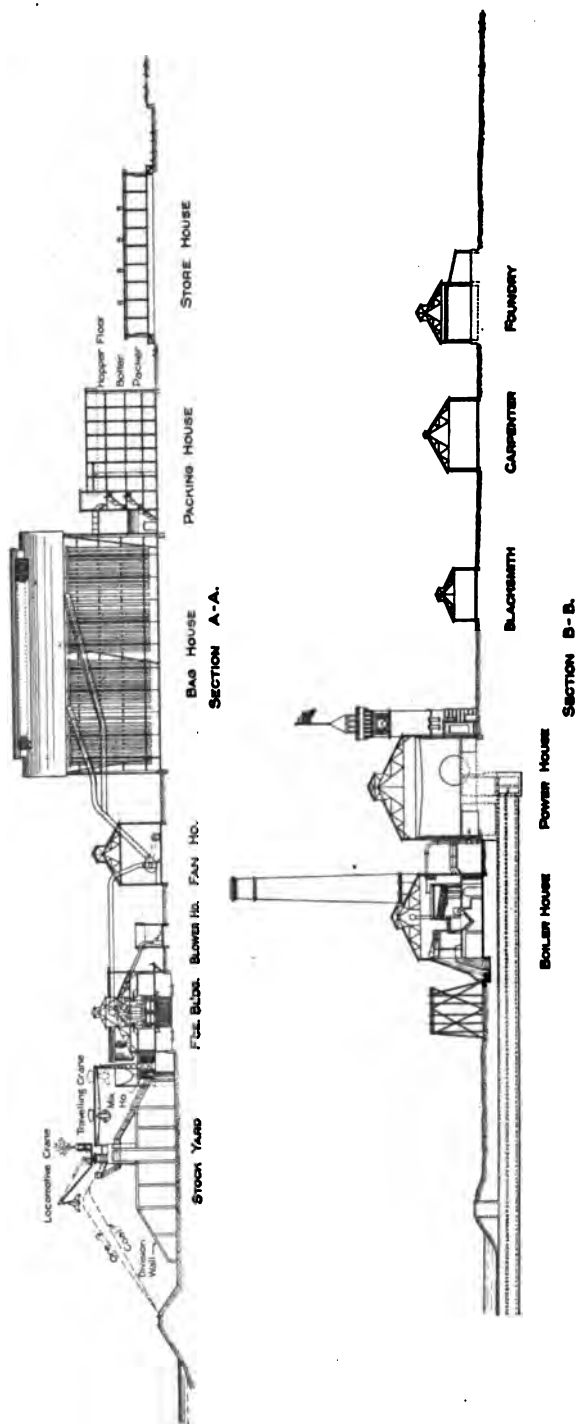


FIG. 4.—SECTIONS OF WEST PLANT.

directly into the bins and the surplus is stocked back of the trestle by a bridge crane of 233 ft. 9 in. (71.25 m.) span equipped with a 10-ton Brown grab-bucket. This equipment also returns materials from the stock piles to the bins when necessary. The charges are drawn from the bins to a weighing larry which has two hoppers, each containing a furnace charge. These are carried to Ransome concrete mixers placed at one end of the furnace room. The mixed charges are elevated by a pair of counterbalanced skips to bins above the furnaces. From the bins the mixed charge is taken to the furnaces by cranes each having three hoppers holding one-third of a furnace charge each. Each hopper is placed on a scale so that the charge is equally divided between the three.

### *Stocking and Mixing—West Plant*

All materials are delivered on a single track steel and concrete trestle 47 ft. (14.4 m.) high, down the center of the storage yard. The materials are taken from this by a bridge crane of 48 ft. 8 ½ in. (14.85 m.) span with a 2-yd. (1.8 m.) bucket and placed in elevated bins next the furnace room. The ore and coal are stocked and reclaimed by the traveling bridge and by locomotive cranes on the high trestle. The total storage capacity is 200,000 tons of ore and 175,000 tons of coal. The fuel and the various ores are taken by bridge cranes to small bins placed over the mixers. The materials are dropped from these bins into weighing hoppers and from these into the Ransome mixers. The mixed charges are elevated to bins over the furnaces and then distributed by cranes much as at the East plant, except that there are no scales on the cranes or larries. This arrangement of doing all of the weighing at the mixer on a fixed hopper is more satisfactory than weighing larries, because the latter are very difficult to keep in adjustment.

### *Furnace Rooms*

The furnace rooms at the two plants are quite similar, the main differences are that at the West plant the furnaces are carried on a concrete foundation and at the East plant, on the floor. In both cases the furnaces are elevated sufficiently to allow the residue to be dropped into hoppers below the floor and then into cars. The framing of the buildings is quite different, the East plant (the later one) being arranged to give more room for the charge cranes. The West plant furnace room contains 34 blocks of furnaces and the East plant, 26. All the blocks are alike, consisting of four furnaces 19 ft. 6½ in. by 5 ft. 11½ in. (5.96 by 1.82 m.) (Fig. 5). Each furnace has independent blast and flue connections and three charge openings in the roof surmounted by hoppers. Each furnace works three charges in 24 hr. The different furnaces of the

block are charged alternately at 2-hr. intervals in regular order. Charging in this way, there is always one furnace starting, one nearly finishing, and two working strongly, and the volume and temperatures of the gases to be handled are nearly uniform at all times.

### *Furnace Operation*

When the furnace is worked off the dampers in the flues are closed, cutting it off from the bag room. The blast is shut off and the working doors taken down. Three men work on the furnace at once, each at a separate door. The loose material on top of the charge is raked off and dropped on the floor close to the furnace. The clinker is then broken up by heavy slice bars and raked into hoppers under the floor.

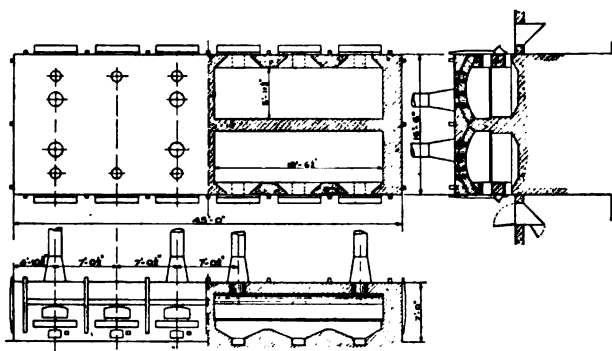


FIG. 5.—FURNACES.

The furnacemen then thoroughly clean from the side and back walls any clinker that has fused to them. When all of the old charge has been removed and the furnace cleaned, the grate is covered as rapidly as possible with a thin layer of coal, the doors closed and a light blast turned on. The coal is lighted by the radiation from the hot arch and it is important that it should light quickly and evenly. When the bed coal is burning brightly, the charge from the hopper is dropped on to it and leveled. The doors are now closed, the blast turned on, and the furnace left to itself with occasional inspection to stop any blow holes that may form. With the large charges that are now worked, it is considerable time before any zinc vapor appears. When it does, the dampers in the flues are reversed, sending the gases and oxide to the bag room.

### *Fans and Piping*

The blowers for the oxide furnaces are direct-connected to motors, the wheels of the blower being placed on extensions of the motor shaft. They are located in small houses outside the furnace room, two in each house.

Each furnace has two outlet pipes, usually 20 to 22 in. (0.5 to 0.56 m.) in diameter which lead up to the drum pipe overhead. The drum has an outlet in the middle of its length leading to the exhaust fans. Formerly the gases passed through a brick settling tower between the furnaces and exhaust fan, but this is now usually omitted as unnecessary. The exhaust fans are extra heavy plate fans with special Hyatt roller bearings and water-cooled shafts, belted to motors. Aside from the shafts, bearings and extra thickness of plate, they are of the ordinary types. For convenience in attendance, a number are placed in one building. The West plant has four fan rooms, three containing eight fans each and one containing ten fans. At the East plant there are two with ten fans each and one with six. From the exhaust fans to the bag rooms, in each case, run round pipes 3 ft. (0.92 m.) in diameter, made of No. 16 sheet iron and provided with numerous clean-out openings. In all cases, the pipes are arranged to give as nearly as possible the same length from furnaces to bag rooms. This is done partly to equalize the friction in the different lines, but mainly to secure proper cooling of the gases before reaching the bag room.

### *Bag Rooms*

The method of collecting oxide in bags was first patented by S. T. Jones in 1852 and has been patented by others many times since. The bag rooms at Palmerton vary considerably in size; the first two built holding bags enough for four blocks, while the latest contains the bags for twenty. The larger rooms are more economical in labor and do not cause any inconvenience, provided the ventilation is equally efficient. The bags used are all of heavy, closely woven cotton sheeting 22 in. (0.56 m.) in diameter and about 40 ft. (12 m.) long. In the majority of cases the gases enter a header at the top of the room and from there pass to a series of parallel pipes from the bottom of which the bags are connected to hoppers, four bags to each, with a single collecting bag tied to an outlet at the bottom of each hopper. At intervals the gases are shut off from the two lines of pipe connecting with a single line of hoppers and the bags shaken. This pair of pipes is again connected to the main header and the next two lines shut off and shaken. The frequency with which the bags are shaken depends upon the relative area of grate and bags, and the amount and quality of the oxide being made. The proper ratio of grate surface to bag surface has been a subject of much discussion. It varies in practice between 1:80 and 1:130. The amount actually necessary depends mainly on the efficiency of the ventilation of the buildings and the frequency of shaking. A well-ventilated room will consume no more muslin with a ratio of 1:80 than a poorly ventilated one with a ratio of 1:130. Recent experiments indicate the probability that a well-



ventilated bag room with a mechanical shaker operating at short intervals will allow of a very great reduction in the necessary bag area.

The collecting bags are removed from the hoppers every 24 hr., or oftener if necessary, and taken to the packing room. This is done by a trucking gang that removes the filled bags and replaces them with fresh ones, trucking them by hand to one side of the bag room where they are coupled together and the train drawn to the packing room by a motor.

### *Packing*

The packing room is a four-story building of mill construction. The trucks are hauled into the ground floor and then taken one by one on a platform elevator to the upper floor. There are a number of openings in this floor leading to bolters on the floor below. Around each opening is a wide shelf on which the bags are rested while their contents are emptied into the bolters. The third floor contains nothing but the bolters, and a room for sewing and making bags. The bolters used are of the ordinary types used in flour mills. The packing machines are arranged in four lines on the second floor. They are also of a type largely used for flour. There is usually one packer below each bolter and fed directly by it. In some cases, owing to the lack of room, one bolter feeds two packers. The oxide is packed partly in paper bags holding 50 lb. (22.7 kg.) each, partly in barrels with 30-in. (0.76-m.) staves and 19 $\frac{1}{8}$ -in. (0.49-m.) heads holding 300 lb. (136 kg.) each, while export material is packed in special barrels and weights to suit the different markets.

The barrels are kept in a storage room where the hoops are driven by a machine and then nailed by a second one; and from here they are passed to the packers. All barrels are weighed empty and filled, the weights adjusted and the barrels headed. They are passed to the ends of the building and rolled on skids to the storehouses. The second floor of the packing room is high enough above the ground level to allow sufficient slope to the skids to carry the barrels to all but the most distant store houses. The paper bags are loaded on trucks and taken to the storehouses on the same skidways.

### *Cooperage*

The cooperage is at the East plant, next the packing room. It is fully equipped with machinery and has a capacity of about 2000 bbl. a day. The barrels are delivered directly from it to the East plant packing room and by special railroad cars to the West plant storehouse.

The storehouses are all one-story mill-construction buildings with the floor elevated to the level of a box car. Each has doors and platforms on opposite sides, one for receiving material from the packing room and the other for loading cars.

*Power Plant*

Power for both plants is supplied from a central station at the West plant between the oxide and the spiegel furnaces. The boiler room is equipped with Edgemoor boilers of 536 hp. each, which can be fired with coal or with the waste gases of the spiegel furnaces. The engine room parallels the boiler room and is located about 20 ft. (6 m.) north of it. It contains three horizontal cross-compound blowing engines for the spiegel furnaces; four direct-current generators, two of 500 and two of 600 kw., each direct-connected to a horizontal cross-compound Corliss engine; three Allis Chalmers turbo-generators, two of 2000 and one of 4000 kw., 6600 volts, and three phases; two motor-generator sets used as a reserve for both alternating- and direct-current lines, as either part can be used as a motor to generate current of the other kind, if needed. In addition, the engine room contains the exciters, condensers and pumps. The engine room is entirely above ground, giving a light basement for the pipes and elevating the machinery above danger from floods. This also permits a railroad track to enter at the ground level, so that new machinery or repair parts can be brought into the building, and there lifted from the cars and put in place by a traveling crane that covers the entire space and can lift the heaviest piece of any of the machines. The switchboards are in a tower at the north side of the building where they are visible from all parts of the generator floor, but do not project into the room. The tower gives ample room for leads and anchorages of cables.

*Ores*

The ores used at Palmerton are almost entirely concentrates from the company's mines at Franklin Furnace and at Sterling Hill, N. J., Three of these products are smelted for oxide, franklinite, half/half and dust. The composition of these is about as follows:

|                        | Franklinite | Half/half | Dust |
|------------------------|-------------|-----------|------|
| Zn.....                | 18.2        | 18.4      | 17.6 |
| Fe.....                | 35.0        | 15.2      | 20.1 |
| Mn.....                | 12.8        | 12.2      | 8.9  |
| SiO <sub>2</sub> ..... | 3.6         | 12.6      | 7.2  |

The franklinite is worked by itself and the residue smelted in blast furnaces for the production of spiegel. The half/half and dust are mixed and the residue wasted, as it contains too little Fe and Mn and too much SiO<sub>2</sub> to be a profitable smelting material.

Franklinite is the easiest of these ores to work and gives the best product. The half/half and fines are mixed and worked together. The charges per furnace average:

|                          | Ore   | Coal  |
|--------------------------|-------|-------|
| Franklinite.....         | 2.62T | 1.62T |
| Half/half and fines..... | 1.95  | 1.36  |

The recovery for franklinite averages about 86 per cent. and for the half/half and fines charge about 78 per cent. Both charges and recoveries vary somewhat with changes in the ore and quality of the oxide desired. The recovery and quality of product are controlled by the selection of the ores, ratio between ore and coal and of both to the grate surface, and what is of great importance, the balance of blast and exhaust and the proper proportioning of both to the charge being worked. Where leaded ores are worked the charges and recoveries vary much more and average figures are of little value. In general, with such ores the charges are a little lighter and the recoveries lower than for the New Jersey ores. The recoveries of lead are usually rather better than of zinc, roughly as 9 to 8. For estimating purposes the make of oxide is generally taken as equalling the combined  $Zn + Pb$  in the ore, the gain in  $O$  and  $SO_2$  about balancing the losses in  $Zn$  and  $Pb$ .

### *Product*

The product is divided into five grades, two for the use of paint manufacturers, two for rubber, and an off-grade that is reworked either for oxide or spelter. For the paint trade, the grading is entirely by color, samples being daily rubbed down with oil and compared with standards. For the rubber trade, slight variations in color do not make so much difference and the grading in this respect is not so close, but freedom from lead and absence of small hard particles is insisted upon.

The variation in composition of the different grades is very slight, the range of composition of all being nearly the same. The material as shipped, uniformly contains 99 per cent. or over  $ZnO$ , the principal impurities being:  $SO_2$ , 0.25 to 0.33 per cent.;  $H_2O$ , 0.10 to 0.50 per cent.; and  $PbO$ , 0.05 to 0.30 per cent.

*Uses.*—The principal use of zinc oxide is as a paint pigment and for this purpose its purity of color and freedom from discoloration by gases and atmospheric conditions, fineness and uniformity make it particularly useful. As a pigment it has the disadvantage of slow drying and a tendency to become unduly hard in time. The general consensus of opinion among paint manufacturers now is that a mixture of pigments is better than any single one. In these mixtures, zinc oxide is an almost invariable constituent because it prevents chalking and gives a surface that retains its color.

Next in importance is its use in rubber goods. In the manufacture of such products, zinc oxide is used for two purposes. It is used as a pigment to produce white goods, and is excellent for this purpose, provided it is substantially free from lead. If this were the only effect it had it might be replaced by cheaper materials, but it also greatly increases the tensile strength of the resulting material. A mixture of pure rubber and sulphur vulcanized has a tensile strength of about 2000 lb. per square inch

and an elongation of 960 per cent. of its original length. The effect of adding zinc oxide is to increase the tensile strength but diminish the elongation, as shown in Table 1.

TABLE 1

| Per Cent. ZnO | Tensile<br>Lb. per Sq. In. | Elongation,<br>Per Cent. |
|---------------|----------------------------|--------------------------|
| 25            | 2,400                      | 720                      |
| 35            | 2,400                      | 700                      |
| 45            | 2,700                      | 680                      |
| 55            | 2,500                      | 620                      |
| 65            | 2,000                      | 540                      |
| 75            | 1,300                      | 400                      |

It will be seen that up to about 60 per cent. ZnO increases the tensile strength of rubber. No other material tested has this effect.

*Leaded Oxide.*—As the supply of lead-free ores is limited and the demand for ZnO is increasing, much more leaded oxide is being made. The process is the same but the charges are usually lighter and the recoveries lower. In most cases the furnaces used are double ended with one or two working doors at each of the opposite ends. This makes the furnace easier to clean, as there is no back wall, which is the most difficult part from which to cut accretions. On the other hand, this type of furnace cools off more when charging and does not, as a rule, light up as quickly. Which type is better is still an open question, the users of each claiming that theirs is superior. In making leaded oxide, it is customary to have a brick combustion chamber over, or close to, the furnace. This insures a sufficient time of contact at a proper temperature to cause the lead to be converted into basic sulphate. This is essential, as all of the oxides of lead are strongly colored and injure the color of the product, while the basic sulphate is very nearly white and does not have this effect. In many cases the bag rooms of the works making leaded oxide have no overhead distributing pipes, the hoppers being connected and used for this purpose.

Leaded oxide is divided into several grades, depending on the amount of lead it contains. This is always stated as the amount of neutral sulphate ( $\text{PbSO}_4$ ) equivalent to the total lead present although it is always present mainly as a basic sulphate. It forms an excellent material for the manufacture of mixed paints. So far this grade has not proved satisfactory to the rubber manufacturers.

#### *Production*

It is impossible to give the production of oxide of zinc, as the statistics are not published by the Government. Such figures as have been published are not comparable, because the grouping of the different products is not always the same. The increase has been very rapid, especially within the last 3 or 4 years.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 17, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## **Zinc Burning as a Metallurgical Process**

BY W. R. INGALLS

(St. Louis Meeting, October, 1917)

THE manufacture of zinc oxide directly from the ore is one of the most important contributions that America has made to the metallurgy of zinc. Heretofore, this has been done chiefly for the production of zinc and zinc-lead pigments, and the method has been known as the Wetherill process. While that term may properly be applied to the process for pigment manufacture, it must be recognized that the principles of the process are of far wider application; and that there is going to be wider application in the immediate future is more than an expectation. I have in mind the use of the process as a method of igneous concentration.

Igneous concentration is broader in its scope than the Wetherill process, for it may be done in the blast smelting furnace, or in the reverberatory, or by other means. Igneous concentration is a rather clumsy term, and I prefer to introduce the new generic expression of "zinc burning," which is convenient, readily understood and precise, for in all cases the zinc oxide is reduced to metallic form, is volatilized and is then burned to zinc oxide, the latter happening, however, after the zinc has been separated from the residue of the ore.

As previously indicated, zinc burning may be done either in blast or reverberatory smelting, the gangue of the ore being scorified and drawn off as slag, while other metallic minerals are reduced to matte or metal. Such processes have heretofore been practised on fairly large scales. Thus, F. L. Bartlett smelted zinc ores at Canyon City, Colo., in specially designed blast furnaces, wherein the smelting column was only about 18 in. high. Such a limitation was necessary in order to prevent zinky accretions in the shaft of the furnace, which in course of time would have interfered with its operation. In running the Bartlett furnaces it was easy to bar them off, indeed, to poke right down into the smelting zone, and keep things going freely. Some iron blast furnaces in Virginia and the spiegeleisen furnaces of the New Jersey Zinc Co. afford other examples of zinc burning in blast-furnace smelting, although in their cases the happening is incidental rather than a primary purpose.

Just as zinc may be burned off in a blast furnace, so also may it be done in a reverberatory. Indeed, there is less trouble, in some respects, in a reverberatory, for therewith there is no shaft in which accretions may

form and hang. The first attempt to burn off zinc in a reverberatory furnace on a large scale whereof I know was made with the Fry process, first in a works at Swansea and later in a large plant on the Manchester Ship Canal. Trial of the process was also made by the Anhaltische Blei- und Silberwerke in Germany. In this process the roasted ore was first smelted with about 25 per cent. of salt cake in a blast furnace, yielding silver-lead and a zinkiferous slag. The slag mixed with 10 to 20 per cent. of coal was then smelted in a reverberatory furnace, wherein the zinc was reduced and burned off. Although something like 25,000 tons of Broken



FIG. 1.—INTERIOR VIEW OF FURNACE BUILDING, EAST PLANT, NEW JERSEY ZINC CO., PALMERTON, PA.

Hill ore was treated by this method, the process did not prove a commercial success. About the same time Ellershausen was working on similar ideas at Angoulême, in France. He began by burning off zinc in a reverberatory, but he abandoned it in favor of a blast furnace.

About the same time, also, three European metallurgists—Pape, Witter and Babe—were working together on the same problem and took out certain patents in common, but subsequently they separated. Pape developed a method of burning zinc out of the Oker slags, developing a special furnace of the blast-furnace order, and made a great commercial success of it, his zinc fume being a large source of zinc supply to the smelt-ery near Hamburg. Babe did further experimental work with apparatus conforming to the principles of the Bessemer converter. Witter took up the idea of reverberatory smelting and about 1908 carried on some rather

extensive experiments at a works in Germany. However, neither Babe nor Witter developed a commercial method. It remained for Frederick Laist, at Anaconda, Mont., to make zinc burning in the reverberatory furnace a successful large-scale process. This was done early in 1917 for the extraction of the zinc remaining in the residues of the leaching for electrolytic zinc extraction.

The blast furnace and the reverberatory furnace are both smelting furnaces. A smelting furnace may be defined as one in which reduction, fusion and scorification are performed. The Wetherill-grate furnace is a blast furnace in which reduction is performed but no fusion and no scorification, or but to a slight degree. There is no run of slag, but merely enough formation of it to clinker the residuum. Anything more would clog the grate and the furnace could not be worked. We have here a



FIG. 2.—FLUE, COOLING PIPES AND BAG-HOUSE PLANT OF ANACONDA COPPER MINING CO. AT GREAT FALLS, MONT.

combination of the conditions that exist in the retort for zinc distillation and in the blast smelting furnace. The zinc is distilled and the residue remains in somewhat the same physical condition as in the former. The product cannot be metallic zinc, however, for just as the carbon dioxide and excess of air act oxidizingly in the blast furnace they act more intensively in the Wetherill furnace. That the reduction of zinc oxide to zinc takes place is well known for the beautiful bluish-green flames of burning zinc may be observed immediately above the charge, but it is quickly converted into zinc oxide by the carbon dioxide and excess of oxygen that are present and what leaves the furnace is gas laden with zinc oxide (and lead oxide and sulphate if lead be present in the ore).

Heretofore the process of zinc burning on Wetherill grates has been confined to the manufacture of pigment. In that are involved problems in collecting a product of the requisite physical properties, especially

purity of color. It is necessary to effect a perfect combustion of coal dust mechanically carried over, a perfect separation of all mechanical impurities, and the ore that is treated must be free from certain impurities, such as cadmium, which affect adversely the color of the zinc oxide. In accomplishing these things much skill, that can be gained only from practice, is necessary. As a means of concentrating zinc, however, no great skill is required, the color of the product being no consideration, and the process then becoming so simple attention is being increasingly directed to it. There are several directions for its application, the most important of which seem to be as follows:

1. As a simple means of concentrating low-grade ore, especially calamine, to save freight and treatment charges on worthless gangue. Thus, the calamine of Leadville might be taken to Canyon City, Colo., the zinc might there be burned off and the zinc oxide shipped to distillers or to electrolytic refiners. This is in fact being done on a small scale.

2. The residues from lixiviation in electrolytic zinc works, containing a large quantity of zinc owing to the formation of insoluble zinc ferrite in roasting, may be burned on Wetherill grates for the extraction of that zinc. This is going to be done. An ore containing 48 per cent. of zinc and yielding 83 per cent. of it to sulphuric acid may have a residue assaying 20 per cent. zinc, the proportion of residue to original ore (roasted) being about one-third, and that 20 per cent. of zinc is well worth extraction.

3. It is possible that it may be found most economical in the electrolytic process to roast the ore, burn it on Wetherill grates and leach the fume.

4. The production of zinc fume in this way affords an excellent supply of zinc for the manufacture of lithopone and other chemical purposes.

All of the above are important opportunities for the process of zinc burning and within the next few years extensive applications of it are going to be witnessed. As to whether the application will be by means of the Wetherill grate or of the reverberatory smelting furnace (I think the blast furnace may be dismissed from consideration) will depend upon the nature of the gangue, the precious-metal content of the ore, and the commercial and metallurgical conditions that prevail at any given place. Before undertaking to analyze the factors that will govern, let attention be directed to the behavior of silver and lead. Gold may be left out of consideration, being of but rare occurrence in zinc ores.

Lead is even more easily burned out of an ore either in reverberatory furnace or Wetherill than is zinc. In my own experience, the extraction of lead has almost always been higher. In treating a lead-zinc ore, therefore, the product will inevitably be a lead-zinc fume. If the ore that is being burned contains sulphides there will be sulphur in the fume, for lead sulphide is volatile as such. The behavior of silver is variable.



With some ores the larger proportion will go into the fume. With other ores it may remain in the residue. It would be preferable if it would all stay in the residue or all go into the fume, *i.e.*, preferable in consideration of subsequent steps for its extraction, but that never is to be expected. As to what determines the behavior of silver I am not prepared to pronounce. I imagine that if the ore contains a little copper, enough to form a matte as collecting agent, more silver will remain in the residue than otherwise. Anyway, that is the experience in blast-furnace smelting.

As between Wetherill-grate burning and reverberatory-furnace smelting and burning, I think it impossible to lay down any hard and fast rules. The subject has been insufficiently studied and analyzed. Reverberatory smelting follows the same lines as for the treatment of copper ore, but sufficient carbon is mixed with the charge to reduce the zinc oxide. No great excess over the theoretical quantity is needed, the conditions being very different from those of the retort for distillation. Nor does it appear that the quantity of coal that must be burned in the firebox is greatly in excess of what is used in smelting an ordinary copper ore. It might be feared that the loss of zinc by scorification would be high in this process. It is rather hard to keep zinc out of the slag in any kind of smelting. Even in electrothermic smelting it is common to have 5 or 6 per cent. of zinc in the slag, with good work, while the percentage may be very much higher. If, then, in order to make a proper slag for the running of the furnace, it is necessary to flux the ore so that the proportion of slag to ore is high, the zinc loss in the slag may be uncomfortably large although the assay of the slag may be moderately low. However, by maintenance of proper conditions, especially a neutral or reducing atmosphere, in the furnace, the zinc tenor of the slag may be reduced to rather surprisingly low figures and the extraction of zinc as fume may compare very favorably with what would be obtained in burning on Wetherill grates.

Bearing on this subject, the *Metalbank und Metallurgische Gesellschaft* in German patent, No. 290,013, Oct. 8, 1913 (addition to No. 252,195) says that the driving off of the zinc from the mixture of ore and fuel has been found to proceed more satisfactorily the smaller the excess of air in the heating gases entering the reaction chamber. Ordinarily in smelting a charge a portion of the zinc is held back by the slag which forms, while the reduced zinc vapor is reoxidized by the excess air to ZnO, which is dissolved by the molten slag. In order to overcome this difficulty, the process should be carried out in a reducing atmosphere, avoiding fusion. In order that the highest possible temperatures may be used, the charge is mixed with ores or additions so that only a sintering results, without fusion. The CO<sub>2</sub> present in the heating gases has been found sufficient to oxidize the zinc vapors without excess of air.

W. Troeller, in German patent No. 291, 853 (Apr. 2, 1913) reports

that experiments have shown that the expulsion of zinc and other metals from liquid slags or melts is dependent upon the degree of oxidation of various metals contained in the slags, *e.g.*, iron, and that such expulsion is practically complete only when these metals have reached their lowest degree of oxidation or are maintained in that condition. In view of this fact, a current of reducing gas, *e.g.*, illuminating gas, water gas, or the like, may be conducted through the slag or melt of ores, maintained in a liquid state at the reaction temperature in a directly or indirectly heated furnace, whereby volatile metals such as zinc, bismuth, antimony, etc., are reduced from their compounds and are driven out with the escaping gases.

In other words, once the oxide of zinc is reduced, the zinc vapor must be given the best possible chance to be volatilized and be reoxidized above the slag bath, not in it. If slagging be prevented, the scorification of zinc oxide will be reduced. It should be theoretically possible, for this reason, to burn the zinc more completely out of an ore on a Wetherill grate than in a reverberatory furnace, and I believe the results of practice are in line with this idea, but, as I have previously indicated, good work with the reverberatory furnace may discharge slag with only 3 per cent. Zn and that is not far behind the zinc assay of the cinder from the best grate burning.

In the use of labor and fuel, the large reverberatory, of the Anaconda type, is far ahead of the best form of the Wetherill furnace. With a small reverberatory the difference will be less, and indeed may disappear. Local conditions will, no doubt, determine the choice of method in all cases. For a small plant to treat ore giving an essentially basic or essentially acid residuum of no value for silver or copper content, the Wetherill furnace would be on the most advantageous terms. If, however, the residuum approximated self-fluxing character and contained silver and copper contents worth recovering by smelting, it would be likely to be most advantageous to smelt at once and be done with it, recovering the zinc incidentally. The latter process will of course suggest itself in connection with the metallurgical treatment of deposits of zinkiferous pyrites containing sufficient copper for matte formation.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS  
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Mining Methods of the American Zinc Co. of Tennessee

BY H. A. COY\* AND H. B. HENEGAR,† B. S., MASCOT, TENN.

(St. Louis Meeting, October, 1917)

THE Mascot mines of the American Zinc Co. of Tennessee are situated in the Holston River valley, in Knox County, Tennessee, about 13 miles (20.9 km.) east of the city of Knoxville, and form a property of three operating shafts and one open pit. A second open pit is located at Jefferson City about 16 miles (25.8 km.) east of the Mascot mines.

### *Geology*

*Topography.*—The principal topographical features of the region within which the zinc deposits occur is a series of parallel northeast and southwest ridges which reach altitudes of 200 to 400 ft. (61 to 122 m.) above the valley. These ridges are formed of resistant formation such as sandstone, quartzites, etc., while the valleys are occupied principally by limestone, including Knox dolomite, and by shale and other formations subject to quick erosion.

The Holston River valley is wide and open with a rolling surface. Sink holes are common to the limestone portion, the general aspect of the valley being that of well-rounded hills with low intervening valleys superimposed upon which is a characteristic sink-hole topography.

*General Geology.*—According to Keith, the rock formation of the area extends from the lower Cambrian into the Silurian. The region has undergone severe deformation with heavy faulting, the underlying Cambrian having been brought to the surface by a series of low angle thrust faults, the strike of the faults and formations paralleling the general northeast and southwest direction of the topography and dipping at varying angles to the southeast. In general the Cambrian quartzites, sandstone, etc., occupy the ridges while the Cambro-Silurian dolomites, limestone and shales occur principally in the valleys. Of the latter, the Knox dolomite is of particular interest to this paper in that it is the ore carrier in the Mascot area. The ore is associated with more or less irregular breccia zones, the degree of brecciation varying from a fine crushing to a breaking that involves blocks of large size. The breccia zones

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grade laterally and vertically into unfractured or slightly fractured ground.

The pay ore so far mined has a maximum thickness of 110 ft. (33.5 m.) and pinches down, or runs into low grade as unfractured portions of the formation are approached.

*Ore Occurrence.*—The ore occurs both as a sulphide and carbonate, the sulphide alone being mined by the American Zinc Co. The carbonate is found only in small pockets at shallow depths.

### *Mining Methods*

Two of the present operating shafts of the American Zinc Co. of Tennessee were sunk approximately 100 ft. (30.5 m.) south of the orebody and are on an east and west line 2800 ft. apart.

No. 1 shaft (which was the first shaft sunk by the American Zinc Co.) is located on what is known as Holston Hill. This is a three-compartment shaft consisting of two skipways 5 ft. 6 in. (1.68 m.) in the clear and one 3 ft. 6 in. (1.07 m.) cageway which has in connection two 24-in. (0.61 m.) pipeways. This shaft is square setted to solid rock ("or bottom of cribbing") with 6 by 6-in. (152.4 by 152.4-mm.) oak timbers. From this point down 8 by 8-in. (203.2 by 203.2-mm.) timbers were used, each set being blocked to the walls, and bearing sets being inserted about every 50 ft. (15 m.); 4 by 6-in. (101.6 by 152.4-mm.) dressed-pine guides are used on the skipways and 4 by 4-in. on the cageways. The guides are joined together by tongue and groove and fastened to dividers with 8-in. lag screws, the heads of which are countersunk in guides. This shaft was sunk to a total depth of 380 ft., the main station being cut at 280 ft. down. From this station a main level crosscut was driven due north, through the orebody to a point under an incline leading from an old shaft which had previously been sunk by former operators. A connection was made between main level and incline by a raise, this being done for ventilation and to give a separate manway to surface, which is one of the requirements of the mining laws of Tennessee. Drifts were driven east and west from main level following foot wall and hanging wall. At 100-ft. intervals along these drifts crosscuts were driven due north, a number of which extended as far as 300 ft. into the foot wall. At points along each crosscut, raises are driven to orebody, the spacing of these raises depending entirely upon the system laid out in advance for mining that particular ground, as each stope is more or less of a problem of its own, and has to be mined independently, especially the stopes on the upper runs which have no connection with the main run.

Several systems have been tried for mining the ore lying above the main level. The shrinkage system was tried on the east side of this mine but did not work out because the ground broke into large slabs and

choked the pull holes. Again, it was almost impossible to get a safe roof for machine men to work under and the conditions as a whole justified discontinuance of this system entirely. •

The underhand system now being worked has proven to be the most satisfactory way of working this ground. With this system, an 8-ft. (2.5-m.) heading is cut along the hanging wall and the roof made safe before the bench is removed. As these headings advance, sublevel trams are established, the ore being dumped into raises which were driven in advance as stated above. As a rule, one raise takes care of

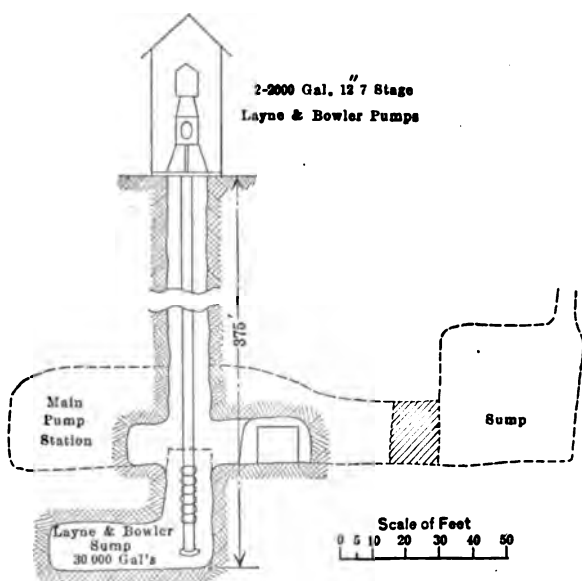


FIG. 1.—SECTION THROUGH EMERGENCY PUMP SHAFT, No. 1 MINE.

two sublevel trams which enter at different elevations. Where there is a higher run of ore, these raises are extended and the same systems worked as on the lower runs. For entering the headings of the lower runs of ore the open stopes are used, but for the upper runs a protected manway with staggered ladderways is furnished, which also takes care of the air and water lines. Where the ore goes below the main level, inclines are put down on the foot-wall side and after cutting at lower stations, drifts are driven east and west and the same stoping system worked as on the upper ore.

No. 2 shaft is located 2800 ft. (853 m.) due east of No. 1. This was the second shaft sunk by the American Zinc Co. and is a four-compartment shaft, consisting of two skipways and one cageway, each 5 ft. by 5 ft. 6 in. (1.53 by 1.68 m.) in the clear, and one pipe and manway 3 ft. by 5 ft. 6 in. (0.92 by 1.68 m.) in the clear. This shaft is square setted

practically the same as No. 1 with the exceptions of 8 by 8-in. (203.2 by 203.2-mm.) oak timber being used for the entire distance. No. 2 shaft was sunk to a depth of 612 ft. (186.5 m.), the main station being cut on the west side at 520 ft. (158.5 m.) down. From this station a haulage drift encircles the shaft and is widened out on the north side where there are two short 8 by 12-ft. (2.5 by 3.7-m.) drifts, one of which goes due east to the head of the east incline, the other due west to the foot of the west incline. The east incline dips  $20^{\circ}$  for a distance of 350 ft. (106 m.). The west incline rises  $24^{\circ}$  for a distance of 825 ft.

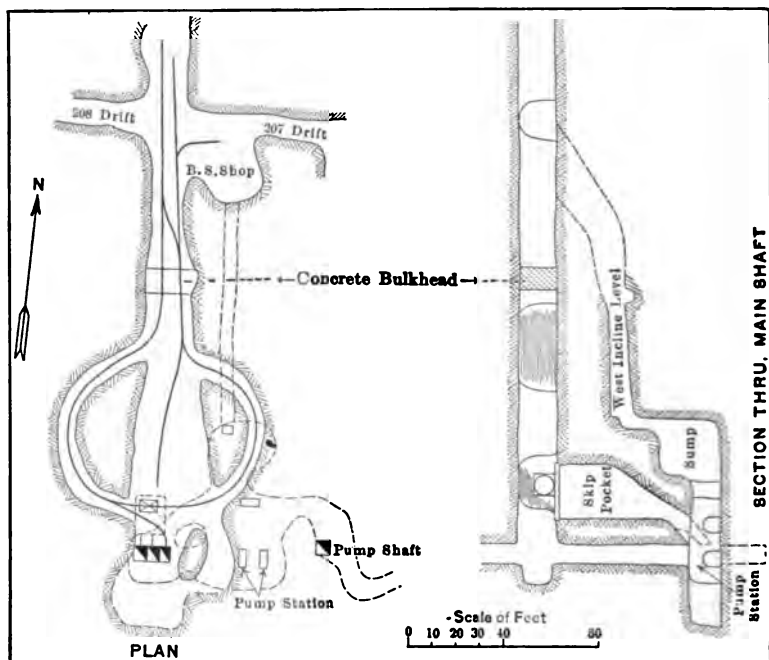


FIG. 2.—PLAN OF MAIN STATION, NO. 1 MINE.

(251.5 m.). Both inclines are cut 8 ft. high and 10 ft. wide and carry a single haulage track of 40-lb. (18.2-kg.) rail, 24-in. gage, which is laid close to the south rib, giving ample room on the north side for a cleated board walkway.

The same general stoping systems are used at this mine as at No. 1, but for handling the ore above the main level the incline system is used in place of the sublevel system. At approximately 100-ft. (30-m.) intervals along both the east and west inclines, crosscuts are driven from the incline on the hanging-wall side north to the foot wall. Each crosscut is protected for tramming and stopes are opened up on the hanging-wall side of each crosscut, working in the direction of the next crosscut

which is approximately 30 ft. (9 m.) higher vertically. After the top heading is cut along the hanging wall, the bench is removed as fast as possible in order to get shoveling room on the crosscut level. The ore on the main level between the east and west inclines is removed by first cutting out the sill floor, then working the overhead system, holding the muck until the roof is made safe and a top heading is cut on hanging wall.

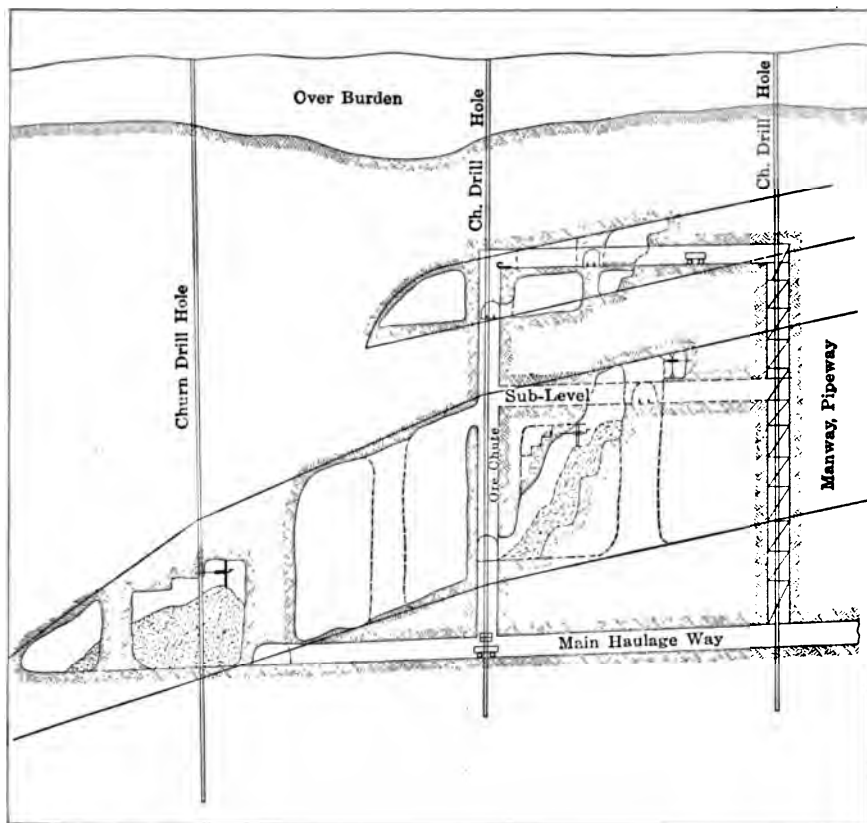


FIG. 3.—SECTION SHOWING SILL FLOOR AND SUB-LEVEL SYSTEMS OF STOPING.

No. 3 mine, which is approximately 3000 ft. (914 m.) due west of No. 1, was formerly worked by the Grasselli Chemical Co. and purchased by the American Zinc Co. in 1914. Work was at once started in making this shaft two compartments 5 ft. by 3 ft. 10 in. in the clear. Quite a lot of time was consumed in catching this shaft up, as it had caved in several years before. After this was accomplished, the shaft was square-setted and concreted to solid rock a distance of 50 ft. From this point 8 by 8-in. oak stulls were inserted in well-cut hitches. These stulls

were placed 6 ft. apart and were for the purpose of carrying in 4 by 6-in. guides.

The former operators had mined approximately 50,000 tons which had been taken out of the immediate vicinity of the shaft at a point 187

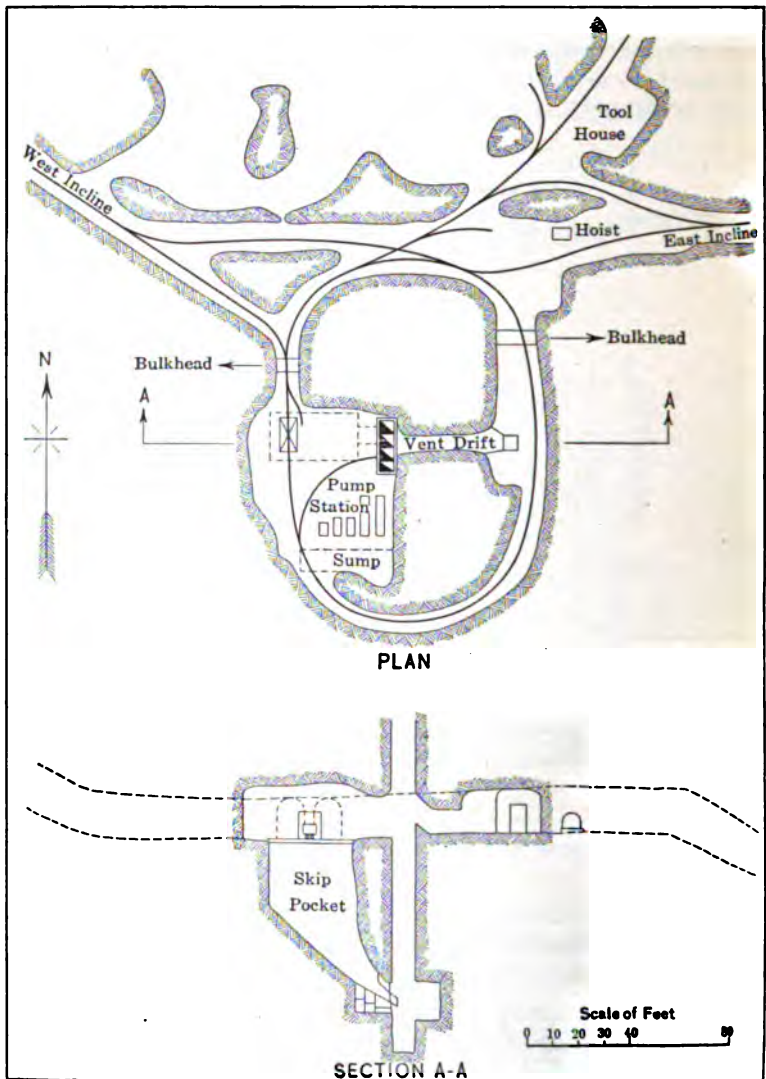


FIG. 4.—No. 2 STATION LAYOUT.

ft. (56.9 m.) below the collar. Previous to the purchase of this property, the American Zinc Co. drove two prospect drifts and one raise. One of these drifts was driven with the ore due east for a distance of 300 ft.



(91 m.), the other crosscutting the ore due north 100 ft. (30.5 m.); from the end of this north crosscut a raise was driven through the ore a distance of 90 ft. (27.5 m.). The general line of development was carried on from these prospect drifts after the mine was purchased. The east drift was extended northeast under open pit and an incline raise connection was made for handling this ore through the mine to the mill. From the raise off the north crosscut a drift was driven due east on the hanging wall of the upper run of ore and stopes opened up at intervals along this drift, the same heading and bench system of mining worked as at No. 1, using the sublevel tram system for handling the ore. The east drift on

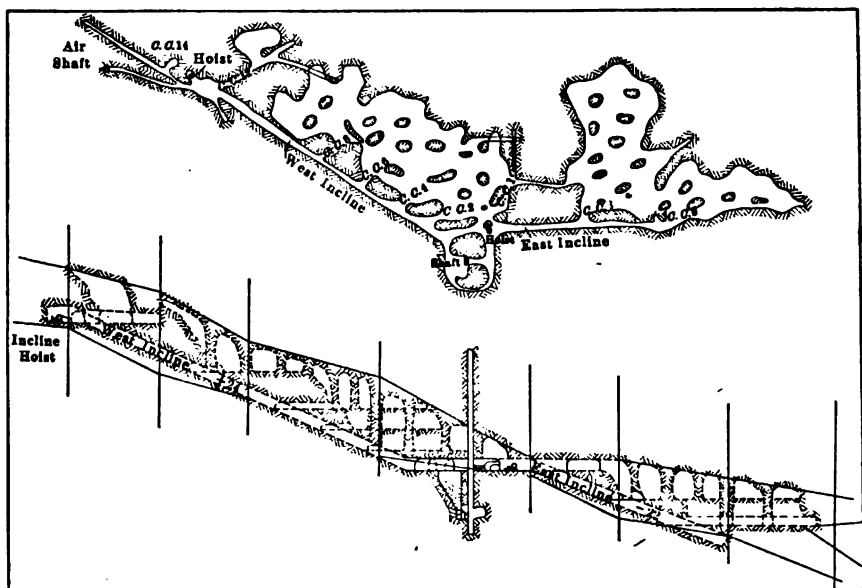


FIG. 5.—PLAN AND SECTION SHOWING INCLINE SYSTEM OF MINING AT NO. 2 MINE.

the main level being driven in the middle of the ore, stopes were opened up on both the north and south side, and after cutting out the sill floor the overhead system of stoping was worked. In developing the ore below the main level, a  $27^\circ$  incline was driven a distance of 200 ft. following the foot wall. From this incline a drift was cut on west side approximately 95 ft. down. At the foot of the incline a foot-wall drift was driven west, which will eventually tap an orebody of 75,000 tons recently drilled out approximately 800 ft. west of this mine. On the east side, at foot of incline, a crosscut was driven to the hanging wall. This hanging wall was then followed a short distance and stopes opened up on the north side.

### *Ore Breaking*

The drilling practice and equipment is the same throughout all the properties. Hammer drills are being used throughout. For heading and drifting work the 18-A Leyner and DR-6 Sullivan mounted drills are used.

For bench work, employing vertical holes, the Leyner machine taken off the shell and with improvised handles bolted to the backhead is used chiefly. The Dreadnaught machine is used for some classes of work. Some stope work is done with the Leyner machines on tripods, where long flat holes are required.



FIG. 6.—PILLAR WORK IN NO. 1 MINE.

This combination of drills for different classes of work is very desirable on account of using one kind of drill steel entirely, thus simplifying the question of sharpening.

The drill steel is  $1\frac{1}{4}$  in. (31.75 mm.) (hollow round) with Leyner shank and Carr bit. The starters are made  $2\frac{1}{4}$ -in. (57.15-mm.) gage with 24-in. (60.96-mm.) changes and  $\frac{1}{8}$  in. (3.175 mm.) is allowed in gage on each change. Runs are made to 14 ft. (4 m.), but holes are rarely drilled deeper than 10 ft. (3 m.).

Leyner-Ingersoll drill sharpeners are used in all the shops. Both No. 3 and No. 5 models are in operation. The shops at mines No. 1 and No.

2 are underground and oil forges have been put into service. The shop at mine No. 3 is on the surface. All steel-sharpening shops are equipped with adequate slack pots, oil-tempering tanks, emery wheels for dressing shanks and individual steel rocks for each working place, from which the steel is checked out and in each shift.

For driving raises the Ingersoll-Rand BC-21 stoper is employed, using 1-in. (25.4-mm.) cruciform steel with the cross bit.

For shaft sinking, winze work, light stoping and blocking, the Ingersoll-Rand BCR-43 jackhammer is used with  $\frac{7}{8}$ -in. (22-mm.) hollow hexagon steel. This steel has a collar shank. Both cross bits and Carr bits are used.



FIG. 7.—MAIN-LEVEL HAULAGE WAY, NO. 1 MINE.

An air pressure of 80 lb. (36 kg.) is maintained at the drills and water is supplied for wet drilling in order to keep down dust.

The ore breaking is all done on company time and each round is personally inspected by a shift boss to determine grade of ore, condition of roof, position of pillars, etc.

The development work, including drifts, crosscuts, raises, winzes, inclines and shafts are commonly done on contract. These contracts are let on basis of feet advanced and contractor furnishes labor and explosives.

The loading is done on company time on a task basis of so many cars per man depending on conditions in each working place.

### *Blasting*

Forty per cent. gelatin powder in  $1\frac{1}{4}$  by 8-in. sticks is used for all blasting. Exploders are made up with Crescent fuse and No. 6 caps.

Electric detonators have been tried out but the cap-and-fuse method has been found more desirable because a majority of holes are dry.

A separate crew of powdermen supervises the stope blasting, which is all done at the end of the night shift.

Underground magazines are provided to accommodate a ton or more of powder, but only one day's supply, or approximately half that amount, is kept on hand.



FIG. 8.—A STOPE IN NO. 2 MINE

### *Tramming*

The sublevel tramming is done by hand with  $1\frac{1}{4}$ -ton end-dump cars. Two men load a certain task and tram each car to a raise. A maximum tram of 300 ft. is maintained.

The main-level haulage is done with 2-ton cars handled by 3-ton storage-battery locomotives. These locomotives travel from 4 to 6 miles per hour and handle 5- to 8-car trips to the tippie.

The tipples in use are of the drum type, motor-driven. One or two cars are run into the tippie and are dumped by being turned a complete revolution about the long axis.

The ore is dumped into a skip pocket having a capacity of 600 tons, and is drawn out through gates operated pneumatically, into 4-ton skips

working in balance. The skips dump automatically into a feeder above the gyratory crusher.

The man cages are double-decked and inclosed with chains or telescope gates. Counterweights are provided so that the cages also work in balance.

All cage and skip hoists are motor-driven and connected with herringbone gears.

### *Mine Drainage and Pumping*

Due to the fact that the orebody lies very close to the Holston River, and in places underneath creeks tributary to it, the question of water is of greatest importance. No. 1 mine has been flooded on two occasions and No. 2 mine had two lower levels under water at one time. This question in itself could be made the subject of a paper, but it is sufficient to say that there are installed at the three mines pumps sufficient to handle 17,000 gal. per minute and pumping plants under construction for an additional 10,000 gal. per minute.

None of the mines are directly connected underground at this time, but because of fissures and caves in the rock, the water during flood times drains from one to the other, making it necessary to have concrete bulkheads and very large pump installations.

The high-speed centrifugal pump is largely used, although plunger pumps of the horizontal duplex and vertical triplex type comprise part of the equipment. All the pumps are motor-driven, either direct- or gear-connected.

### *Power*

Electrical power is obtained from the Tennessee Power Co.'s hydroelectric plant at Parksville. This plant is located in Polk County 100 miles from Mascot, and a 66,000-volt transmission line connects the two places.

The Zinc company has an emergency power plant consisting of a 1000-hp. steam turbine sufficient to carry the pumping load at any time the power is out of commission.

The power plant for compressed air includes two Ingersoll-Rand 1600-cu. ft. per minute compressors and one Nordberg 3500-cu. ft. per minute compressor, all being direct-connected to synchronous motors.

### *Typical Crew*

Table 1 shows distribution of the crew for a mine producing 1300 to 1350 tons per day.

All loading and tramming is done by negro labor; drilling and miscellaneous work by native white labor.

TABLE 1

|                         | Day Shift | Night Shift |
|-------------------------|-----------|-------------|
| Shift boss.....         | 1         | 1           |
| Shoveler bosses.....    | 2         | 2           |
| Machine men.....        | 18        | 16          |
| Machine helpers.....    | 18        | 16          |
| Blacksmiths.....        | 2         | 2           |
| Shovelers.....          | 44        | 42          |
| Chute pullers.....      | 6         | 6           |
| Powdermen.....          | ..        | 4           |
| Pipemen.....            | 4         | ..          |
| Roofmen.....            | 5         | ..          |
| Timbermen.....          | 2         | ..          |
| Car repairmen.....      | 2         | ..          |
| Tool boys.....          | 1         | 1           |
| Cage men.....           | 1         | 1           |
| Trackmen.....           | 4         | 2           |
| Motormen.....           | 2         | 2           |
| Motor couplers.....     | 2         | 2           |
| Incline hoistermen..... | 2         | 2           |
| Incline couplers.....   | 4         | 4           |
| Tipplemen.....          | 1         | 1           |
| Skip loaders.....       | 2         | 2           |
| Total.....              | 123       | 106         |

On a tons-per-man basis, the following is a fair example of results obtained:

15.7 tons per shoveler; 5.7 tons per man (total underground).

The pumping crew is not included in any of the above figures.

TABLE 2.—Costs

|                                    | Per Ton        |
|------------------------------------|----------------|
| Drilling and boulder breaking..... | \$0.220        |
| Blasting.....                      | 0.110          |
| Roof protection.....               | 0.015          |
| Loading.....                       | 0.150          |
| Haulage and car repair.....        | 0.080          |
| Hoisting and skip loading.....     | 0.040          |
| Tracking.....                      | 0.015          |
| Foremen and miscellaneous.....     | 0.070          |
|                                    | <b>\$0.700</b> |

This includes about 40 ft. of development per week, which represents \$0.04 of the breaking cost. Pumping expense is quite variable, but under normal conditions amounts to \$0.03 per ton of ore handled and is not included in the above.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Development and Underground Mining Practice in the Joplin District

BY H. I. YOUNG,\* CARTERVILLE, MO.

(St. Louis Meeting, October, 1917)

INASMUCH as there has been a great deal of activity in this district recently, a paper of this kind should treat of all the various phases of mining, namely, prospecting, developing and operating.

### PROSPECTING

Where ore is found at a depth of 100 ft. (30 m.) or more, the prospect work is done by churn drilling. Several makes of churn drills are used in this work, both steam- and gasoline-driven machines being used. The drilling is usually started with a 6-in. bit, and the size of the bit is reduced, if necessary, as the hole is advanced. It is general practice to drill from 100- to 400-ft. holes, but this is governed entirely by the location and formation of the ground. In drilling sheet ground, close spacing of drill holes is unnecessary, but in narrow orebodies, the holes are spaced from 15 to 50 ft. (4.6 to 15 m.) apart. Ten years ago, very few assays were kept of the drilling, but during recent years records have been kept very accurately, showing the location, elevation, assays, and also the vertical section of the hole.

After the orebody is reached, cuttings are taken every 2 or 3 ft. (0.6 to 0.9 m.) samples being sometimes taken by pouring the water off the cuttings and taking a portion of the coarse material for assay. It has been found, however, that very often the sludge material poured off with the water contains high mineral values. To overcome this, cuttings from each 2 or 3 ft. are put in a container and after the water is evaporated, sample for assay is taken by means of Jones samplers.

Permanent records of cuttings showing various formations are kept in glass jars or glued on large cards. Drilling at present is costing from \$1.25 to \$1.50 per foot. In pre-war times, the drilling costs ran from 75c. to \$1 per foot.

### DEVELOPMENT

After proving property by drilling, shafts are put down to the orebody. These are usually vertical shafts, either single or double compart-

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\* Manager, Missouri Mines, American Zinc, Lead and Smelting Co.

ment. The dimensions of the one-compartment shafts vary usually from 4 by 5 ft. (1.22 by 1.52 m.) to 5 by 7 ft. (1.52 by 2.2 m.) in the clear, while those of the two-compartment shafts run from 5 by 10 ft. to 7 by 12 ft. in the clear. Recently a few three-compartment shafts have been sunk. The shafts are usually put down near the edge of the orebody, so that very little drifting is required to open up the ground for operating. The shafts are usually cribbed with 2 by 4-in. or 2 by 6-in. (50.8 by 101.6 mm. or 50.8 by 152.4 mm.) sawed timbers (very few square sets being used because of the nature of the ground), and extend several feet below the top of the solid rock. The cribbing is lined or laced vertically with 1-in. boards, which help to make the shaft free from small rocks and water, and permit faster hoisting. The cost of sinking shafts varies according to the nature of the ground, the amount of water encountered, and the size and depth of the shaft. The average cost of a 5 by 7-ft. shaft, at present, where from 100 to 500 gal. (379 to 1893 l.) of water per minute is handled, to a depth of from 200 to 250 ft. (60.9 to 76 m.) is from \$25 to \$30 per foot. Usually two or more shafts are sunk on each operation. These shafts are connected with an air drift 5 by 7 ft. which gives good ventilation. The cost of driving a drift varies from \$6 to \$9 per foot.

In the Granby, Aurora and Galena camps, where ore is found at from 40 to 100 ft. (12 to 30.5 m.) the ordinary practice of developing ground is by sinking small shafts; very little drilling is done in proving up the shallow orebodies.

A good way to increase tonnage where working faces are limited is by driving a 7 by 8-ft. drift ahead of the working faces to a distance of 300 to 600 ft. and opening up the ground so as to permit the use of several machines. Drill holes are usually necessary for proper ventilation where this method is used.

## OPERATING

### *General Practice*

The methods employed in the underground work vary with the nature of the deposits, and have been developed to suit the local conditions encountered in the various mines. The ore deposits lie in a horizontal plane, and are usually found on one level, varying in heights from 8 to 40 ft. (2.5 to 12 m.). In some parts of the district, however, two distinct levels are found and in a few instances as many as four separate ore strata have been found, these being separated by a relatively thick stratum of hard, barren rock, and the several runs are mined independently of each other.

In the mines where the orebody is higher than 16 ft. (4.9 m.) "underhand stoping" is practiced, especially in the hard-ore sheet-ground mines. Pillars are left at frequent intervals, the distance between pillars and



thickness depending on the nature of the ground and the height of the orebody. The ore is usually found in hard flint formation, known as the "Grand Falls Chert," which lies between beds of limestone. The lime

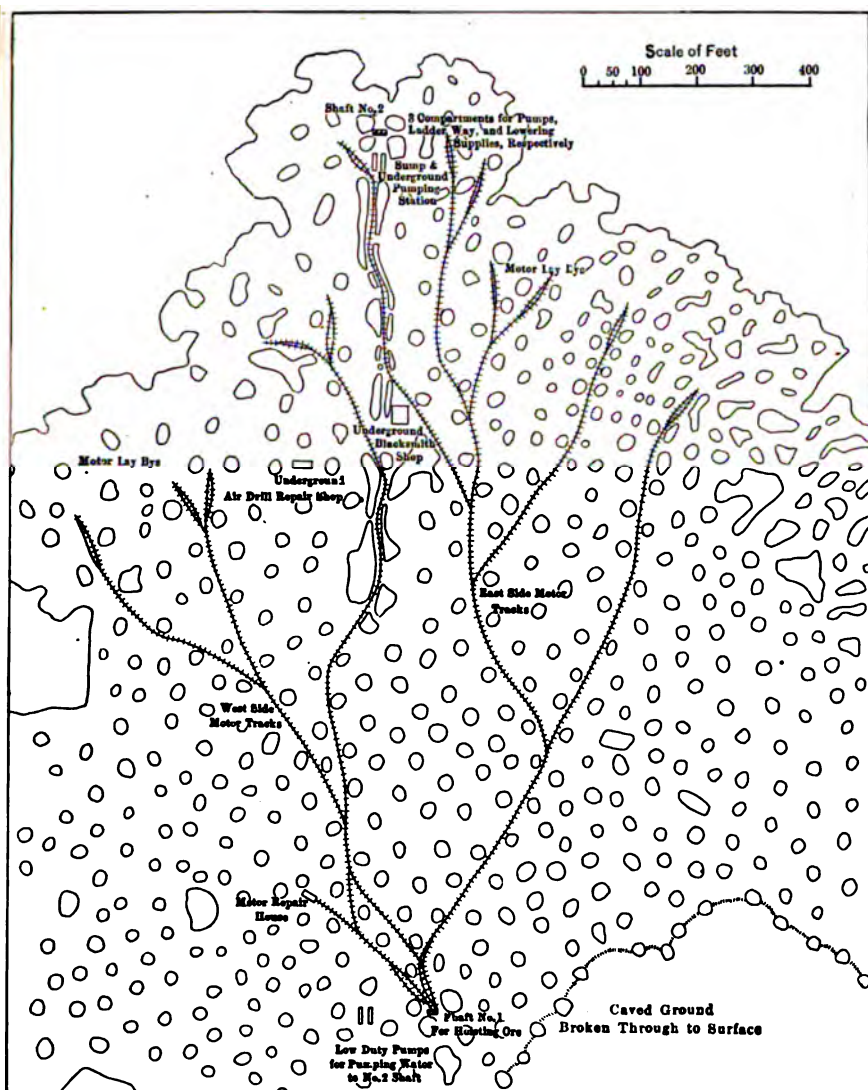


FIG. 1.—UNDERGROUND WORKINGS OF A SHEET-GROUND MINE.

formation is known as the "Mississippian," and is overlaid by the "Pennsylvanian" series of shales and sandstones.

A comparatively small amount of timbering is required in the mining operations of the district, as there is a cap rock of flint varying in thick-

ness from 1 to 5 ft. (0.3 to 1.5 m.), which makes a good roof in the majority of mines. In some cases, this flint decomposes into what is commonly known as "cotton rock." This is soft and white, and where it occurs pillars must be placed very closely, and the roof arched. In sheet-ground from 10 to 20 per cent. of the ore formation is left in the form of pillars. These are sometimes trimmed, and many of them removed after the ore-body has been worked out. Fig. 1 shows the pillar system in a typical sheet-ground mine.

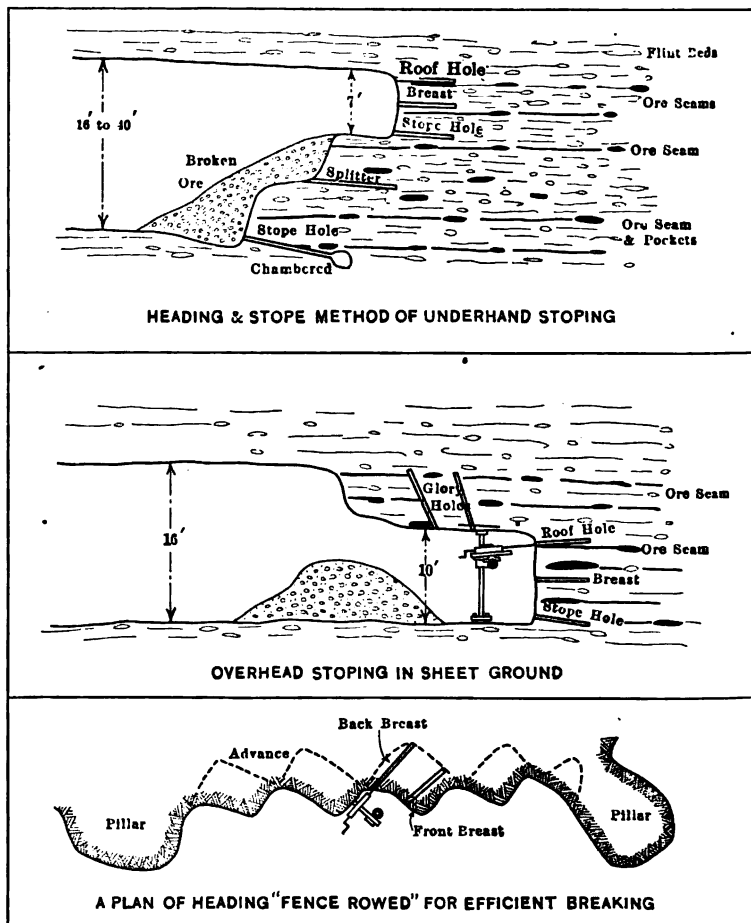


FIG. 2.

### *Drilling and Breaking*

Where the orebody is over 16 ft. in height, it is customary to employ the "heading and stope" method of cutting the ground. The heading is carried on 6- or 7-ft. posts. The face is very irregular, which is necessary

for the best breaking results. From five to seven holes are drilled at each set up, and the heading is usually advanced from 15 to 20 ft. ahead of the stope. If the stope is over 10 ft. in height, a splitter is drilled horizontally so as to relieve the load on the stope hole. In cases of very high stopes, more than one splitter is used. Stope holes are drilled so as to dip below the horizontal in order to maintain a level bottom.

Where the orebodies are less than 16 ft. in thickness, the "heading and overhead stoping" method is profitably employed. A 10-ft. post is used to drive a bottom heading, from five to eight holes being drilled to each round. These holes vary in depth from 10 to 12 ft., and usually from 75 to 150 tons of ore are broken to each round of holes. After the bottom heading is advanced from 10 to 25 ft., holes are drilled in the overhanging ore, and this is blasted down at a very low breaking cost.

A comparison of these two methods is shown in Fig. 2.

Two types of air drills are used in this district, the "piston and hammer" types. In sheet-ground mines, where the laying of dust is an important consideration to the health of the miners, the "hammer" type or water drill has been used with great success, it has also shown an increased drilling efficiency of from 25 to 40 per cent. above the "piston" machine. Drills are operated by compressed air, at a pressure of from 85 to 100 lb. per square inch.

High-carbon hollow drill steel is used with "hammer drills." Holes are started with a  $2\frac{3}{4}$ -in. bit and finished with  $1\frac{1}{4}$ -in. It is necessary to change the size of gage in each 2 ft. drilled, and it is sometimes necessary to use several steels to drill 2 ft., the ground being exceedingly hard. The drill steel is resharpened at an underground blacksmith shop, or on the surface where oil forges have been found to give the best results. Because of the large amount of steel used, it is absolutely necessary to sharpen the steel by mechanical sharpeners.

Drilling costs vary with the character of ore mined. The tabulation in Table 1 shows costs for 4 months of 1917, in a mine where the faces are from 10 to 16 ft. in height.

TABLE 1

| Tonnage broken.....136,272  |             |                  |
|-----------------------------|-------------|------------------|
|                             | Amount      | Cost Per Ton Ore |
| Machine men.....            | \$9,630.40  | \$0.0706         |
| Machine helpers.....        | 7,940.45    | 0.0583           |
| Drill repairs.....          | 3,505.14    | 0.0257           |
| Drill steel.....            | 2,627.29    | 0.0193           |
| Sharpening drill steel..... | 3,194.86    | 0.0234           |
| Oil for machines.....       | 207.78      | 0.0015           |
| Compressed air.....         | 6,168.63    | 0.0453           |
| Air hose and fittings.....  | 703.43      | 0.0052           |
| Total.....                  | \$33,977.98 | \$0.2493         |

*Blasting*

After drilling, the holes are chambered and prepared for powder. Both ammonia and gelatin dynamite of a strength of from 30 to 40 per cent. is used, in the form of sticks of from 1 to  $1\frac{1}{4}$  in. diameter. No. 8 caps have been found to give the best results. The powder crew in mines where hoisting is carried on double shifts do their work on the second shift, and fire shots when the shift goes off; this gives the mine several hours to be cleared of powder smoke and gases. It is necessary to have skilled labor for this work, both on account of the high cost of explosives, and for the safety of the men. The holes are loaded with charges varying from 25 to 150 lb., depending upon the formation of the ground, and the burden of the hole.

Breaking, or blasting, costs are from 15 to 30 c. per ton, varying with different heights of working faces.

The costs shown in Table 2 are for a 12-ft. working face.

TABLE 2

| Tonnage..... | Amount      | Cost Per Ton Ore |
|--------------|-------------|------------------|
|              |             | 136,272          |
| Labor.....   | \$4,006.20  | \$0.0294         |
| Powder.....  | 17,914.00   | 0.1314           |
| Fuse.....    | 782.33      | 0.0057           |
| Caps.....    | 217.62      | 0.0016           |
| Total.....   | \$22,920.15 | \$0.1681         |

These costs are on powder purchased on contract, the present open market price being 70 per cent. higher.

*Shoveling*

After the ore is broken it is sometimes necessary to drill large boulders with a jackhammer, and blast them, to reduce them to shoveling size. The usual practice of the district is to shovel into tubs, which are commonly known as "cans," holding from 800 to 1200 lb. (362.9 to 544 kg.) of ore. These are usually 30 by 30 in., 30 by 32 in., or 32 by 32 in., and are trammed to shaft on small trucks. The shovelers use short-handled No. 2 scoops and an average of 20 tons per shoveler per shift is obtained. All shoveling is done on a contract basis, and shovelers earn from \$3 to \$7 per 8-hr. shift. Both the Thew steam shovel and the Myers-Whaley shoveling machine have been used for underground shoveling to good advantage, but without a decrease in the shoveling cost. However, if labor continues to become scarcer, it may be necessary to adopt this method to insure a steady production.

A number of the mines have adopted the use of cars to take the place of "cans." These hold from 1500 to 2000 lb. (680 to 907 kg.) of ore.

The cars are easier to shovel into, as they are lower than the "cans." They are also more stable, which permits faster tramming.

Shoveling costs vary from 20 to 25 c. per ton of ore.

### *Tramming*

Several different methods of tramming are employed in this field, namely, hand tramming, mule haulage, motor haulage, and rope haulage, depending on the length of the haul, tonnage required and the grade encountered.

The tracks are from 15 to 24-in. gage, the steel varying in weight from 12 to 30 lb. (5.5 to 13.6 kg.) per yard.

The trucks are either wooden or steel frame mounted on plain wheel and axle, or roller-bearing. The roller-bearing truck has been found to make a saving in tramming expense.

Where the haul is 200 ft. (61 m.) or less, the shovelers usually run their cans directly to the shaft, but for any distance above this and up to 400 ft., additional tramming labor is required to take the dirt from the shovelers' lay-by to shaft. Where a large tonnage is required, as is always the case in sheet-ground mines, and the distance from the shaft to the working faces is from 400 to 1500 ft., mule haulage is very successful. A mule will haul 100 tons 1500 ft. in an 8-hr. shift, at a cost of from 5 to 6 c. per ton.

In the large sheet-ground mines, where approximately 1000 tons are hoisted in an 8-hr. shift, and all of this ore must be hauled 2000 ft. or more, motor haulage is necessary. The 6-ton gasoline locomotive is the most popular and gives no trouble with regard to ventilation. Roller-bearing cars are required for long hauls.

Rope haulage is used in some mines where the grade is very severe.

The costs in Table 3 show comparison of gasoline locomotive and mule haulage for 4 months of 1917. The distance on locomotive haulage is 1750 ft.; on mule haulage, 700 ft.

TABLE 3

| Gasoline-locomotive Haulage |            |              | Mule Haulage           |            |              |
|-----------------------------|------------|--------------|------------------------|------------|--------------|
| Tons hauled.....136,272     |            |              | Tons Hauled.....81,145 |            |              |
|                             | Amount     | Cost Per Ton |                        | Amount     | Cost Per Ton |
| Engineer.....               | \$1,883.62 | 0.0138       | Mule driver.....       | 1,868.70   | 0.0230       |
| Brakeman.....               | 1,659.12   | 0.0122       | Maintenance mules      | 406.68     | 0.0051       |
| Oil for locomotive...       | 239.27     | 0.0018       | Car couplers.....      | 964.56     | 0.0119       |
| Gasoline for loco-          |            |              |                        |            |              |
| motive.....                 | 1,458.10   | 0.0107       | Car greaser.....       | 470.37     | 0.0058       |
| Repairs.....                | 2,063.39   | 0.0151       |                        |            |              |
| Car couplers.....           | 2,241.65   | 0.0164       |                        |            |              |
| Total.....                  | \$9,545.15 | \$0.0700     |                        | \$3,710.31 | \$0.0458     |

*Hoisting*

Three systems of hoisting are used in this district; cans, cages and skips. The can is the best adapted to this depth of hoisting, and the tonnage requirement of the average Joplin mill. Both steam and electric hoists are used. The hoisting cycle is as follows: A trammer or bumper runs a loaded can to the side of the platform and the hoisting cable is unhooked from the empty can on the platform and attached to the load by the tub-hooker; this operation is done very quickly and without any signals to the hoisterman, who now picks up the loaded can which swings in over the platform where the tub-hooker steadies it in the center of the shaft. This is raised to the top of the derrick, where the hoist is located. Here the hoisterman hooks a tail rope, which is fastened out in front of the shieve timbers, to a ring in the bottom of the can. The hoisting cable is slacked, and weight is taken by the tail rope, causing can to dump directly into the mill hopper. The hoisting cable is now taken up, and the tail rope unfastened so as to allow the can to descend the shaft. With practice, a hoisterman and tub-hooker will become so expert at their respective operations that as many as 150 cans an hour can be hoisted from a depth of 250 ft. The record of a single compartment shaft for the district is 1071 cans in an 8-hr. shift. Where two compartments are used for hoisting, the hoists are placed side by side in the top of the derrick, and the shaft is divided for its entire depth to prevent cans from bumping.

In order to do all the shoveling on the day shift, and to secure a large tonnage, many of the operators hoist from several shafts, and tram the ore on the surface to the central mill hopper by means of inclined surface trams; some few aerial trams are used, but the level topography and short hauls have made the inclined tram nearly universal in this district.

The cost of hoisting is between 6 and 7 c. per ton, while surface tramming costs 1 c. per ton.

Table 4 shows a distribution of hoisting cost for 4 months of 1917.

TABLE 4

|                              |            |                  |
|------------------------------|------------|------------------|
| Tons hoisted.....            | 136,272    |                  |
|                              | Amount     | Cost per Ton Ore |
| Hoisting engineer.....       | \$2,316.07 | \$0.0170         |
| Tub-hooker.....              | 2,115.57   | 0.0155           |
| Cable.....                   | 720.50     | 0.0052           |
| Hoister repairs.....         | 429.92     | 0.0033           |
| Hoister derrick repairs..... | 110.85     | 0.0008           |
| Shaft lacing repairs.....    | 99.16      | 0.0007           |
| Steam for hoister.....       | 2,744.92   | 0.0202           |
| Slickers for hooker.....     | 140.30     | 0.0010           |
| Oil and waste.....           | 73.16      | 0.0005           |
| Total.....                   | \$8,750.45 | \$0.0642         |

*Pumping*

The serious problem of pumping in the Joplin mines is not the amount of water to be handled, but, in many instances, the acid character of the mine water. It is seldom that over 1000 gal. (3785 l.) per minute has to be pumped at any one mine. In the sheet-ground camp, the mines are so cut together that many of them do no pumping at all; where this is the case, a central drainage company is formed, to do the pumping for all the operators benefited.

Many types of centrifugal pumps are used underground; the Pamoma, and Texas are favorite surface installations, the Texas being used very generally over the district for unwatering mines, while the large-gear duplex or triplex pumps are usually found in the large permanent underground pumping stations.

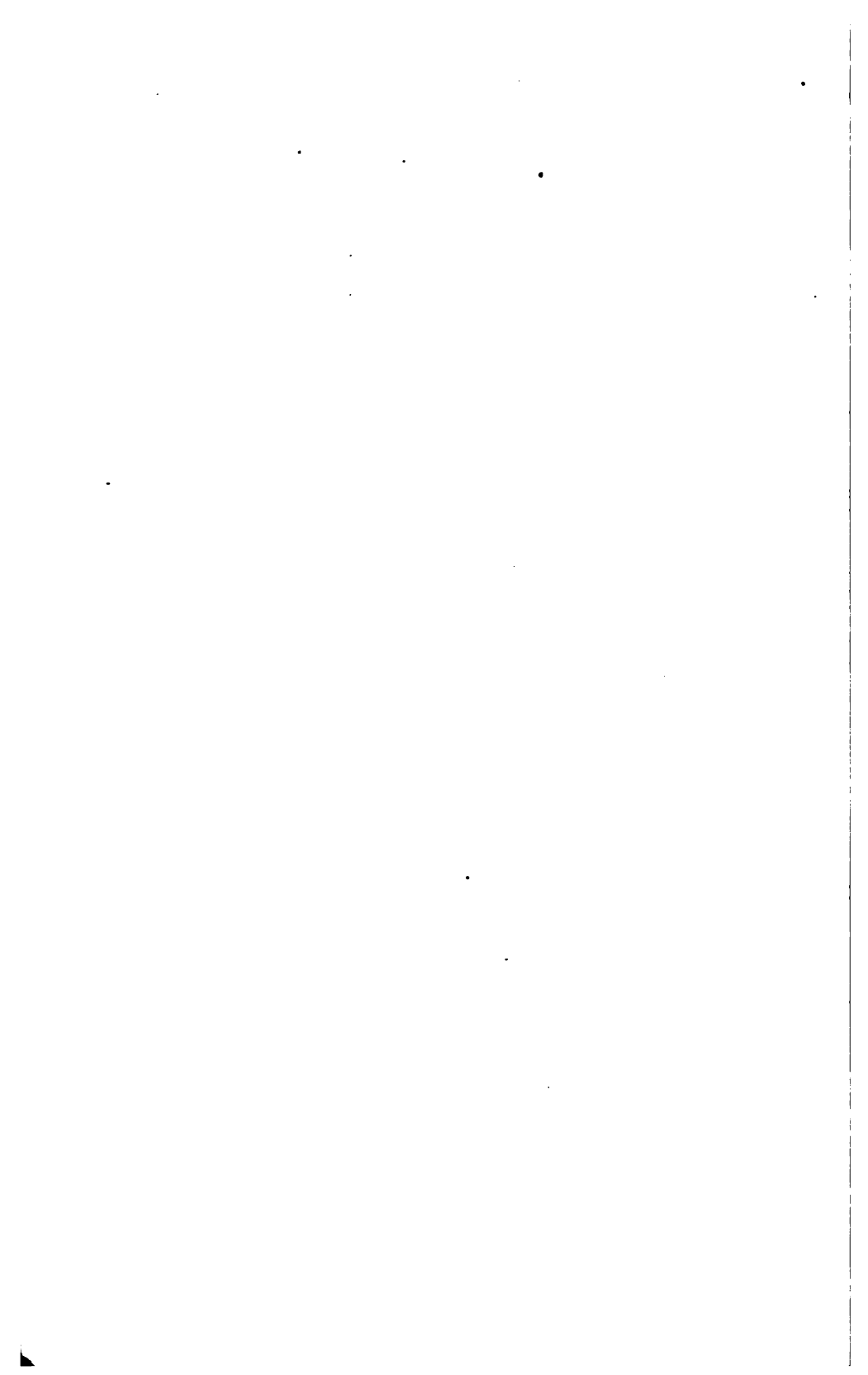
*Costs*

There is no uniform system of cost accounting among the mines of this district. At present, operating costs, which include mining, milling, pumping, miscellaneous and administrative, are from \$1.20 to \$1.75 per ton of ore mined. On account of high prices of supplies and the scarcity and inefficiency of labor, costs are gradually increasing throughout the district.

Table 5 shows a distribution of mining costs over a period of 4 years, 2 before-war years and 2 during the war.

TABLE 5.—*Underground Mining Costs*

| Tons Hoisted            | 1912         |             | 1913         |             | 1915         |             | 1916         |             |
|-------------------------|--------------|-------------|--------------|-------------|--------------|-------------|--------------|-------------|
|                         | 306,263      |             | 303,331      |             | 387,436      |             | 374,937      |             |
|                         | Amount       | Per Ton Ore | Amount       | Per Ton Ore | Amount       | Per Ton Ore | Amount       | Per Ton Ore |
| Ground foremen.....     | \$3,102.50   | \$0.0101    | \$2,942.43   | \$0.0097    | \$4,468.99   | \$0.0115    | \$4,462.99   | \$0.0119    |
| Drilling.....           | 60,886.71    | 0.1988      | 53,671.22    | 0.1769      | 99,824.46    | 0.2576      | 103,800.20   | 0.2769      |
| Blasting.....           | 50,762.53    | 0.1657      | 49,371.41    | 0.1628      | 71,469.12    | 0.1845      | 67,040.58    | 0.1788      |
| Roof protection.....    | 1,857.26     | 0.0061      | 1,341.80     | 0.0044      | 6,169.85     | 0.0159      | 6,817.19     | 0.0182      |
| Shovelling.....         | 47,671.59    | 0.1557      | 44,658.31    | 0.1472      | 81,585.48    | 0.2106      | 87,756.52    | 0.2341      |
| Conveying to shaft..... | 24,434.08    | 0.0798      | 27,880.36    | 0.0919      | 42,301.72    | 0.1092      | 53,045.54    | 0.1415      |
| Hoisting.....           | 14,340.92    | 0.0468      | 13,118.55    | 0.0433      | 15,446.79    | 0.0399      | 18,126.29    | 0.0483      |
| Lighting.....           | 1,767.64     | 0.0058      | 2,113.28     | 0.0070      | 2,389.49     | 0.0062      | 1,929.96     | 0.0051      |
| Miscellaneous.....      | 3,991.05     | 0.0130      | 2,312.82     | 0.0076      | 13,143.44    | 0.0339      | 21,654.02    | 0.0577      |
| Betterments...          | 65.43        | 0.0002      | .....        | .....       | 2,555.54     | 0.0066      | 4,987.64     | 0.0133      |
| Total.....              | \$208,879.71 | \$0.6820    | \$197,410.18 | \$0.6508    | \$339,354.88 | \$0.8769    | \$369,620.93 | \$0.9858    |





DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Zinc Ores of the Joplin District. Their Composition, Character and Variation

BY W. GEO. WARING,\* M. SC., WEBB CITY, MO.

(St. Louis Meeting, October, 1917)

### *Introduction*

THE winning of zinc and lead ores from the comparatively shallow deposits of the Joplin district presents few such problems for the mining engineer as are encountered in deep ore mining and in the handling of complex ores.

However, for those who are interested in the origin and nature of ore deposits in general, there is much here to investigate, and upon studies in this direction will depend in no small measure the future of an industry that has produced during the year 1916, in southwest Missouri, lead and zinc ores to the value of \$26,319,383; adding \$46,022,164 to the metallic wealth of the nation. Including the adjacent mines in Ottawa County, Oklahoma, and Cherokee County, Kansas, which are really continuations of the Joplin district, the total value of ores produced reached the sum of \$34,089,503, and that of the metals, \$58,927,941.<sup>1</sup>

The ores produced in the Joplin district are remarkable on account of their almost uniformly high metal content in both lead and zinc. Their superiority in this regard is unparalleled, and is doubtless due to certain geological conditions that are now possibly explicable, provided that the artesian circulation = ascension hypothesis of the origin of the ore deposits is proven beyond question to be the only tenable explanation of the facts.

These, as well as other valuable zinc-ore deposits, are bound to be exhausted in time, for ordinary mining purposes, and it concerns the mining engineer that the conditions under which ores of this character were actually deposited in the form in which they occur, may, if possible, be ascertained, in order that he may learn where to look for similar deposits in other regions, or even how to reproduce artificially such condi-

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<sup>1</sup> U. S. Geological Survey Bulletin No. 320 (June, 1917).

\* Analytical Chemist

tions as may render available other low-grade sources that cannot be exploited by present methods.

The purpose of this memoir is to contribute certain specific data relative to the character and composition of the ores produced in the Joplin district, in the hope and belief that it may be of service in future studies of the problem just now suggested.

Without further prelude I therefore append here:

1. A table exhibiting variations in the commercial grades of concentrates produced from 34 of the principal mines of the district during 3 months in 1916. This table is only tentative and is incomplete for the reason that few mines keep their first- and second-grade ores separated while very few have all the products assayed for lead and lime and none for cadmium.

2. To remedy in a measure the last-named deficiency, a table giving a series of analyses of concentrates from a number of mines throughout the district showing in particular the variations in the cadmium content of the concentrate. This table shows also the zinc, lead and iron assays and, in 18 cases, the copper. Only the cadmium, however, is an essential or molecular constituent of the sphalerite, the other metals being adventitious; part of the iron coming from the crushing plates and rolls in the mill, the rest, with the copper and lead being derived from the minerals marcasite, chalcopyrite and galenite associated with the sphalerite in the crude ore or "mine dirt."

3. Supplementary analyses, in more or less detail, of typical calamine, blende and lead ores, mine water, leach and flue dust from the calcination of Webb City—Carterville blende concentrates.

The analyses of calamine and mixed blende and calamine show the extent to which they are separated from each other by the ordinary jig process. The calamines here consist almost entirely of the silicate (hemimorphite or true calamine), with very little smithsonite.

The analyses of lead ores (low-grade and average) show the character of the impurities. A considerable portion of the iron abraded from the crushing rolls finds its way into the galena or lead concentrates because it goes through the screen of the first cell of the jig, where all the lead is collected (see footnote to analysis *f*, Table 3).

An analysis of a high-grade, pure honey-yellow, soft flaky sphalerite from the Cottonwood district in Utah, associated with galenite, greenockite, rhodochrosite, calcite, etc., is given, with permission of the U. S. Geological Survey, for comparison with two full analyses of blende concentrates from the Joplin district.

The analysis of acid mine water is typical of the mine waters in certain areas about Galena, Joplin and the Carterville districts. The mine waters of some other areas are alkaline. The leach water from tailing dumps was collected in April, 1917, after a rain.



TABLE 2.—*The Relative Content in Zinc, Cadmium, Copper, Lead and Iron of Various Zinc Ore Concentrates (Including One Sample of Rich Crude Ore or "Mine Dirt") as Shipped from the Joplin District, 1902-1916*

| Name of Mine                        | Locality               | Description             | Per Cent., Zn | Per Cent., Cd | Per Cent., Cu | Per Cent., Pb | Per Cent., Fe |
|-------------------------------------|------------------------|-------------------------|---------------|---------------|---------------|---------------|---------------|
| Standard.....                       | Fortuna, Mo.           | Rosin jack              | 61.97         | 0.436         | 0.133         | 0.815         | 0.55          |
| Standard and Gundling (2 cars)..... | Fortuna, Mo.           | Rosin jack              | 63.50         | 0.530         | 0.107         | 0.323         | 0.90          |
| Big Six (3 cars)                    | Aurora, Mo.            | White jack              | 56.75         | 0.018         | 0.004         | None          | 1.88          |
| Uncle Sam....                       | Aurora, Mo.            | White jack              | 61.20         | 0.690         | 0.014         | None          | 0.75          |
| Ayres and Messick.....              | Stratford, Mo.         | Lead gray, pebble jack  | 64.05         | 0.841         | 0.015         | 0.625         | 0.80          |
| Hudson.....                         | Pleasant Valley, Mo.   | Ruby red pebble jack    | 62.05         | 0.322         | 0.030         | None          | 0.61          |
| Jack Rose.....                      | Alba, Mo.              | Yellow blende           | 54.70         | 0.225         | .....         | 1.025         | 1.38          |
| Sphinx.....                         | Neck City, Mo.         | Yellow blende.          | 65.77         | 0.135         | 0.077         | None          | 0.55          |
| Willard.....                        | Neck City, Mo.....     | Yellow blende.          | 63.30         | 0.320         | 0.064         | 0.410         | 0.80          |
| Big Circle.....                     | Oronogo, Mo.           | Yellow blende           | 56.90         | 0.110         | Tr.           | 1.510         | 1.64          |
| Oronogo Mutual                      | Oronogo, Mo.           | Yellow blende           | 61.87         | 0.410         | 0.018         | 0.265         | 1.38          |
| McKinley.....                       | Carterville-Prosperity | Dark brown rosin blende | 57.20         | 0.550         | None          | 5.290         | 1.25          |
| Lucky Budge..                       | Carterville-Prosperity | Dark brown rosin blende | 59.90         | 0.380         | .....         | 0.990         | 1.06          |
| Welton.....                         | Carterville-Prosperity | Dark brown rosin blende | 57.40         | 0.410         | 0.036         | 1.340         | 3.05          |
| Holy Smoke....                      | Carterville-Prosperity | Dark brown rosin blende | 60.13         | None          | 0.070         | 0.102         | 1.20          |
| Ramage No. 1.                       | Carterville-Prosperity | Dark brown rosin blende | 63.60         | 0.410         | 0.018         | 0.265         | 1.38          |
| Twin Cities....                     | Carterville-Prosperity | Dark brown rosin blende | 46.00         | 0.270         | .....         | 0.600         | 8.60          |
| McDonald.....                       | Carterville-Prosperity | Dark brown rosin blende | 57.27         | 0.277         | 0.030         | 1.500         | 1.56          |
| Edgar Zinc Co.                      | Carterville-Prosperity | Dark brown rosin blende | 61.30         | 0.300         | .....         | 0.134         | 1.00          |
| Edgar Zinc Co.                      | Carterville-Prosperity | Dark brown rosin blende | 62.45         | 0.490         | .....         | 0.003         | 0.90          |
| Edgar Zinc Co.                      | Carterville-Prosperity | Dark brown rosin blende | 59.60         | 0.200         | .....         | 0.033         | 1.32          |
| Munson (Continental)                | Carterville-Prosperity | Dark brown rosin blende | 59.20         | 0.068         | 0.054         | 0.293         | 1.90          |
| Maude B.....                        | Webb City              | Rosin jack              | 55.70         | 0.227         | Tr.           | Tr.           | 4.90          |
| October.....                        | Webb City              | Rosin jack              | 58.10         | 0.460         | None          | 0.590         | 2.74          |
| Majestic.....                       | Webb City              | Rosin jack              | 57.05         | 0.330         | None          | 1.610         | 1.82          |
| Three Shaft....                     | Webb City              | Rosin jack              | 60.20         | 0.140         | 0.086         | 0.887         | 2.08          |
| Bertha A.....                       | Webb City              | Rosin jack              | 59.80         | 0.220         | .....         | 0.650         | 3.40          |
| Wingfield.....                      | Webb City              | Rosin jack              | 60.00         | 0.375         | .....         | 0.600         | 3.00          |
| Little Butler...                    | Webb City              | Rosin jack              | 56.90         | 0.327         | .....         | 1.610         | 2.30          |
| Incline.....                        | Webb City              | Rosin jack              | 55.60         | 0.083         | .....         | 1.400         | 4.50          |
| White Dog....                       | Webb City              | Rosin jack              | 60.00         | 0.410         | .....         | 1.070         | 1.70          |
| Av. of 5 cars, misc.....            | Webb City              | Rosin jack              | 60.28         | 0.520         | 0.040         | 0.372         | 2.00          |
| Yellow Dog....                      | Webb City              | Rosin jack              | 57.95         | 0.710         | .....         | 1.620         | 1.60          |
| Yellow Dog, "Mine Dirt"             | Webb City              | Crude Ore               | 12.78         | 0.170         | .....         | 0.293         | 1.90          |

The very high average metallic content of the zinc and lead jig concentrates is evident from an inspection of Tables 1 and 2.

These concentrates are produced from a heterogenous mixture of galena and sphalerite, with jasperoid flint and chert gangue, along with clay, shale, limestone and varying amounts of calcite, dolomite, marcasite and chalcopryrite, and in some places calamines and oxidized lead minerals, with no other preparation for the jig process than crushing it, just as it is blasted down in the mine, to pass a  $\frac{3}{8}$  or  $\frac{1}{2}$ -in. mesh screen. It then runs over a roughing jig to carry off the coarser barren matter and

the "smittem" from the "rougher" finally passes over a 5 or 6-cell cleaning jig which yields the clean galena from the first cell, return middlings from the second and clean blende from the remaining cells.

The explanation for the production of such surprising results from a seemingly crude and simple process is to be found, apart from the skill of the jig operators, in the physical condition of the mineral and gangue.

The studies of the ore deposits of the Joplin region, made by Siebenthal<sup>2</sup> during 1902-1914 have shown pretty conclusively that these ores of zinc and lead were most probably deposited during the present geological epoch, from circulating artesian waters of alkaline composition, which had become charged with  $H_2CO_3$ ,  $Si(OH)_4$  and  $H_2S$  during their circulation under pressure through the Cambrian and Ordovician strata, whereby they were fitted to dissolve the metallic sulphides that abound in the rocks of those formations, and to transport them in solution west-

TABLE 3.—*Supplementary Analyses*  
Blende and Calamine Concentrates  
Jig Separation

| Mine                          | Blende | Cal.  | Blende | Cal.  | Cal.  |
|-------------------------------|--------|-------|--------|-------|-------|
|                               | a      | a     | b      | b     | c     |
| Total Zn, per cent.....       | 56.40  | 57.30 | 58.40  | 51.80 | 40.00 |
| Zn as ZnS, per cent.....      | 39.80  | 18.20 | 40.10  | 8.65  | 2.91  |
| Zn as Calamine, per cent..... | 16.60  | 33.10 | 18.30  | 43.15 | 37.09 |
| Fe, per cent.....             | 2.20   | 1.50  | 3.40   | 1.30  | 0.80  |
| CaO, per cent.....            | 0.45   | 0.42  | 0.15   | 0.40  | 4.05  |
| MgO, per cent.....            | Tr     | ..... | Tr     | ..... | 1.44  |
| Pb, per cent.....             | 0.34   | None  | 0.52   | None  | None  |
| Insoluble, per cent.....      | 11.67  |       |        |       |       |

a and b from Wentworth.

c from Princeton.

Lead Concentrate

|                      | d     | e     | f     |
|----------------------|-------|-------|-------|
| Pb, per cent.....    | 74.30 | 80.00 | 77.09 |
| Zn, per cent.....    | 5.20  | 5.16  | 3.05  |
| Fe, per cent.....    | 4.20  | 3.90  | 4.32* |
| Insol. per cent..... | 1.30  | ..... | 0.91  |

d Carbonate ore, Granby.

e Galena, Granby, containing 6.5 per cent. Pb as  $PbCO_3$ .

f Composite sample of concentrate shipped from all parts of district.

\* Metallic Fe = 1.28 per cent. + Fe as  $FeS_2$  = 3.04 per cent.

<sup>2</sup> C. E. Siebenthal: Origin of the Zinc and Lead Deposits of the Joplin Region. *U. S. Geological Survey, Bulletin No. 606* (1915).

TABLE 3.—*Supplementary Analysis.*—(Continued)  
 Blende Concentrate                      Utah Sphalerite

|                                 | <i>g</i> | <i>h</i> |                              | <i>i</i> | <i>j</i> |
|---------------------------------|----------|----------|------------------------------|----------|----------|
| Zn, per cent....                | 58.260   | 59.200   | Zn, per cent...              | 48.200   | 65.020   |
| Cd, per cent....                | 0.304    | 0.058    | Cd, per cent..               | 0.903    | 0.112    |
| Pb, per cent....                | 0.700    | 0.293    | Pb, per cent...              | 11.020   | 0.505    |
| Cu, per cent....                | 0.049    | 0.054    | Cu, per cent....             | 0.220    | 0.440    |
| Fe, per cent.....               | 2.230    | 1.900    | As, per cent...              | Heavy    | 0.013    |
| Mn, per cent....                | 0.010    | None     | Sb, per cent.                | Tr.      | 0.021    |
| CaCO <sub>3</sub> , per cent.   | 1.880    | Tr.      | Fe, per cent...              | 0.970    | 0.032    |
| MgCO <sub>3</sub> , per cent.   | 0.850    | None     | Mn, per cent...              | 1.740    | 0.406    |
| BaSO <sub>4</sub> , per cent.   | 0.820    | Tr.      | Ca, per cent...              | 1.910    | 0.082    |
| SiO <sub>2</sub> , per cent.... | 3.950    | 6.971    | Mg, per cent..               | 0.340    | Tr.      |
| Ni, Co, per cent..              | .....    | Tr.      | Ba, per cent...              | .....    | 0.112    |
| S, per cent.....                | 30.420   | 31.301   | SO <sub>4</sub> , per cent.. | .....    | 0.078    |
|                                 |          |          | S, per cent....              | .....    | 32.120   |
|                                 | 99.773   | 99.777   | CO <sub>2</sub> , per cent.  | .....    | 0.320    |
|                                 |          |          | SiO <sub>2</sub> , per cent. | .....    | 0.287    |
|                                 |          |          |                              |          | 99.548   |

*g* Composite sample of 3800 car shipments from the Joplin district.

*h* A typical sample of sheet-ground blende concentrate from Mine No. 13.

*i* Specimen from Cottonwood district, Utah; composed of honey-yellow sphalerite, with pink spar (rhodochrosite) galena, calcite, etc.

*j* Selected clean honey-yellow sphalerite from *i*.

Flue Dust from the Calcination of Webb City-Carterville Blende Concentrates

|                        | Water Soluble, Per Cent. | Total, Per Cent. |
|------------------------|--------------------------|------------------|
| Zn.....                | 22.526                   | 29.876           |
| Cd.....                | 2.726                    | 2.850            |
| Pb.....                | Tr.                      | 2.661            |
| Cu.....                | .....                    | Tr.              |
| Fe.....                | 0.120                    | 3.650            |
| CaO.....               | .....                    | 1.040            |
| MgO.....               | .....                    | 0.440            |
| SiO <sub>2</sub> ..... | .....                    | 10.264           |
| SO <sub>4</sub> .....  | 37.240                   | 37.240           |
| S.....                 | 0                        | 33.040           |
| Tl.....                | 0.030                    | 0.030            |
|                        | 62.642                   | 91.091           |

wardly until, arriving at localities where structural conditions, namely, the absence of the impervious covering of the Chattanooga shale and the presence of fissures communicating with the overlying subcarboniferous limestone beds, enabled the solution, under heavy pressure, to rise to

TABLE 3.—*Supplementary Analysis.*—(Continued)

Mine Water (A Typical Acid Mine Water from Duenweg)      Surface Water from the Leaching of a Tailing Dump Pile

|                                    | Parts Per Million |                                    | Parts per Million |
|------------------------------------|-------------------|------------------------------------|-------------------|
| Free $\text{H}_2\text{SO}_4$ ..... | 29.40             | Free $\text{H}_2\text{SO}_4$ ..... | None              |
| $\text{FeSO}_4$ .....              | 157.40            | $\text{FeSO}_4$ .....              | 1,033.6           |
| $\text{Fe}_2(\text{SO}_4)_3$ ..... | 179.00            | $\text{Fe}_2(\text{SO}_4)_3$ ..... | Trace             |
| $\text{ZnSO}_4$ .....              | 2,840.50          | $\text{ZnSO}_4$ .....              | 9,092.7           |
| Pb.....                            | 1.90              | Cd.....                            | 3.7               |
| Cd.....                            | 5.25              | Cu.....                            | 3.2               |
| Cu.....                            | 0.30              | Mn, NiCo, etc.....                 | None              |
| Mn.....                            | 24.50             |                                    |                   |
| Ni, Co.....                        | 2.10              |                                    |                   |
| $\text{CaSO}_4$ .....              | Saturated         |                                    |                   |

within say 300 ft. (91 m.) of the surface terrain, where exits to the surface were found at points not completely covered by impervious strata.

At such points the diminution in pressure permitted the gases,  $\text{H}_2\text{S}$  and  $\text{CO}_2$ , to be released, so that the dissolved silica precipitated as a gel, forming amorphous jasperoid when hardened, in place of the limestone taken into solution as bicarbonate, while the sulphides slowly crystallized out, selectively, in the forms in which they are found here. That is, each mineral species was deposited in separate crystals or crystalline masses with little or no interpenetration, the galena usually in the roof of an opening and the sphalerite on the floor. Certain faces of sphalerite masses are well oriented and the chalcopyrite, where it is found, is almost always perfectly oriented throughout.

In that member of the lower Mississippian known as the Grand Falls chert, in the vicinity of Joplin, and in what is known as the Duenweg-Oronogo ore channel, passing between Webb City and Carterville, is found the most productive "run" of the now famous "sheet-ground" deposits. The Grand Falls chert formation consists, where it is not ore bearing, of thin bands of alternating chert and limestone. Where it is ore bearing (as in the Duenweg-Oronogo ore channel, which follows the direction of a nearly vertical fissure 3 or 4 ft. wide, now entirely closed, but perhaps not everywhere, by ribbons or upright layers of jasperoid flint) the limestone bands have been replaced with the sulphides and in places with calcite, and jasperoid, on each side of this fissure for distances varying from several hundred feet to half a mile or more. The wider extensions of the ore-bearing ground appear to coincide with oblique cross-fractures, some of which show slight displacement. The main fissure of this particular channel is well defined in the lower workings of the Oronogo Mutual mine and in the Oronogo Circle mine it is visible almost to the surface. It has

been traced at other points, from Duenweg to near Georgia City, for a distance of 15 miles.

Over the sheet-ground horizon, large cavernous deposits have been mined, which were undoubtedly outlets of the artesian mineral solutions, for they were filled with irregular deposits of blende and galena and other minerals in a gangue of brecciated chert associated with clay and fragmentary débris resulting from the tumbling in of surface strata. These form the "open-ground" or soft-ground deposits, and were the first to be mined in this district.

The conditions which obtain in other portions of the region differ in some details but the hypothesis as to their probable origin applies to all.

The sulphides, then, in these deposits are hard and crystalline, and when they are comminuted, as in the processes of milling, each minute fragment remains approximately an isometric crystal; the galena particle a cube, the sphalerite a hextetrahedron and the chalcopyrite a perfect tetrahedron. The marcasite, being orthorhombic instead of isometric, gives fragments which are sufficiently near to being roughly spherical to answer for practical purposes in concentration.

The chert and jasperoid flint gangue, which makes up more than 90 per cent. of the "mine dirt" in the sheet-ground mines, splits up into conchoidal splinters and flakes that are peculiarly adapted to jig work facilitating a rapid and complete separation of the ore.

Variations in the color and physical appearance of the sphalerite are to be seen which can hardly be accounted for satisfactorily by difference in chemical composition or in the composition or nature of the associated gangue or of the rock inclosing the deposit. Light yellow sphalerites were formerly preferred by ore buyers as being "softer," and therefore more easily calcined and retorted than the supposedly harder red ores or the hard "pebble jack," the name given to small individual hextetrahedrons scattered through clay or other soft matrix.

The ores of the Joplin district differ considerably from those of the Wisconsin region, although their origin is supposed to be of similar nature. The Wisconsin ores contain a notable amount of arsenic and antimony, two elements that have not been found in the Joplin ores. The marcasite is associated with the Wisconsin ores to an extent quite unknown anywhere in the Joplin district. I have no exact data as to the relative proportion of cadmium in the Wisconsin and the Missouri zinc ores, but I am inclined to believe that the ratio of cadmium to zinc is somewhat higher in the Joplin ores than in those of other zinc-producing regions, and particularly in those ores that are recovered from deeper-lying deposits.

Zinc, cadmium and tin, along with the rare metals gallium, indium, thallium and germanium, are closely related in the periodic group system of Mendeléeff.



Leqoc, in 1875, with an assistant, Jungfleisch, extracted 62 grams of the fluid mercury-like metal gallium from 2400 kg. (5291 lb.) of Pierrefitte blende, and had exhibited it in the Paris Exposition of 1878.

In 1915, a very observant watchman at the zinc smelter of the Bartlesville Zinc Co., in Oklahoma, discovered globules like mercury that had oozed out in warm weather upon the drusy surfaces of the cakes of lead residuum that had been taken out of retorts used for the redistillation of crude spelter at a temperature of about 350° C. below the highest temperature attained in the ore distillation. He collected a handful of the liquid metal and submitted it to F. G. McCutcheon, the chief chemist, who found that it gave the reactions of both gallium and indium, and it was later proved to be an alloy of those elements.<sup>3</sup>

In 1916, G. H. Buchanan,<sup>4</sup> chemist of the New Jersey Zinc Co., taking advantage of the volatility of the higher chloride as predicted by Mendeléeff in 1869 for the hypothetical *ekasilicon*, succeeded in proving quantitatively the presence of the extremely rare metal germanium in Joplin ore, as well as in some products of zinc metallurgy.

Still more recently, I have found thallium to be a constituent of zinc ores in the Webb City-Carterville district.<sup>5</sup>

Practical methods for extraction of the rarer metals from zinc-blende ores are as yet lacking. It is necessary that the development of such methods shall be controlled by very delicate spectroscopic tests, requiring the use of the expensive high-grade quartz prisms and lenses, such as are provided in the new Hilger spectrographs, because the important spectral lines of germanium and gallium lie mostly in the ultraviolet. On account of the high temperatures required for the volatilization of these elements, only the spark and arc spectra are available, using resublimed graphite electrodes. A good beginning, however, has been made since Urbain pointed the way several years ago, and a special research is now being considered under the direction of the U. S. Bureau of Standards.

In this connection, it should be noted that Urbain<sup>6</sup> in examining by this method 64 blendes from different localities found that all of them contained Cd and Pb; 59 contained Ga, Ag and Cu; 41 contained In; 38 contained Ge; 32 contained Sn; 26 contained Sb; 10 contained Bi; 9 contained As and 5 contained Mo.

It is also of interest to note the progress and surprising results that are being obtained by the application of a new instrument of research

<sup>3</sup> W. F. Hillebrand and J. A. Scherrer: Recovery of Gallium from Spelter in the United States. *Journal of Industrial and Engineering Chemistry* (1916), 8, 225.

<sup>4</sup> The Occurrence of Germanium in Zinc Materials. *Journal of Industrial and Engineering Chemistry* (1916), 8, 585.

<sup>5</sup> See flue dust analysis.

<sup>6</sup> *Revista real academia cientifica*, Madrid, 8, 49-63.

—the Roentgen rays—to the crystal surfaces of zinc blende, etc., and the analysis of the remarkable interference phenomena which are produced.<sup>7</sup>

To return to the question of extracting the rare elements from zinc ores, I have to suggest that no particular difficulty lies in the way of separating the cadmium, thallium and indium along with copper, etc., in metallic form from the leach liquors of the electrolytic process by cementation upon granulated spelter, while gallium and germanium would doubtless concentrate in the electrolyte until their precipitation by some method yet to be devised could be effected.

The analysis of flue dust from Webb City and Cartersville ore calcination showing an increase in the cadmium-zinc ratio to 15 times that of the original ore, and the conversion of 95.4 per cent. of the cadmium from  $\text{CdS}$  into soluble  $\text{CdSO}_4$ , suggests that the extraction of the cadmium from such material should be made by hot water, followed by cementation upon zinc, instead of returning the flue dust, as is commonly done, to the calcining furnace and ultimately compelling the unwelcome element to enter into the spelter or be dissipated in fume. As about 75.5 per cent. of the zinc in the flue dust is also soluble in water, it should be precipitated by lime after the separation of the cadmium, and the lime-zinc precipitate distilled by a special method for the recovery of the zinc. This has been proven to be both feasible and economical. The recovery of the cadmium from the retort fumes is not difficult, according to methods already in use, but the fumes could also be leached like the flue dust, using an acid solvent, and the cadmium could then be recovered by cementation.

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<sup>7</sup> For a full description of the methods used, and the manner in which the solution of the molecular structure of blende, fluorite, pyrite and calcite is effected by the X-ray spectrometric method, see the memoir of Prof. W. L. Bragg in *Proceedings of the Royal Society of London (A)*, **89**, 468-489.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS  
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 59th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Milling Practice of the St. Joseph Lead Co.

BY L. A. DELANO,\* E. M., BONNE TERRE, MO.

(St. Louis Meeting, October, 1917)

DURING 1916, the St. Joseph Lead Co. milled 2,505,670 tons of ore. This is a daily operating average of 7855 tons. The economic concentration of such a large tonnage necessarily requires a plant equipped with modern machinery and operated efficiently. The milling process used in producing this output has been a gradual development through the long period of years that the company has been in existence.

### HISTORICAL SUMMARY OF MILLING PRACTICE

Ore was first mined at Bonne Terre, Mo., in 1864. The early milling practice was crude. It consisted of breaking the ore into small pieces by hand, pulverizing in a Blake crusher and a pair of Cornish rolls to approximately minus 6 mm., and jigging in the old-fashioned hand jigs. Only the coarse heavy galena was saved, the gangue being shoveled off and discarded. The output for the first year under this method was only 240 tons of pig lead.

The first mill of the company was a combination of extensions and additions built from time to time as the mines developed. It was a museum of early milling machinery, such as jigs, percussion tables, buddles, dolly tubs, log washers, etc., following no definite flow sheet. The chats, or tailings, were hauled to waste dumps by mule and ox teams, as much as 500 tons per day being moved in this manner, until a railroad was built in 1880. This mill was burned in 1883, at which time the annual output had reached approximately 7300 tons of pig lead.

A new mill of 500 tons capacity was designed and built by C. B. Parsons. This mill has been admirably described by H. S. Monroe.<sup>1</sup> The ore was crushed in Blake crushers, Cornish rolls, through 6-mm. screens, and treated by jigs of the Parsons type, percussion tables and log washers, or trunking machines, for cleaning the galena. The middlings were recrushed in rolls and treated on Harz jigs. The mill was operated without change, except later installing Gates crushers and en-

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\* Mill Superintendent, St. Joseph Lead Co.

<sup>1</sup> *Trans.* (1888-89), 17, 659.

larging the capacity to 1200 tons, until 1910, when Wilfley tables replaced those of the Rittinger type.

In 1904, a new mill was built at Leadwood, Mo., with a capacity of 1200 tons daily. The ore was crushed in Gates gyratory crushers and high-speed rolls, and concentrated on Harz jigs and Wilfley tables. In 1906, the Hancock jig was introduced to the district in this mill.

FLOW SHEET  
BONNE TERRE MILL

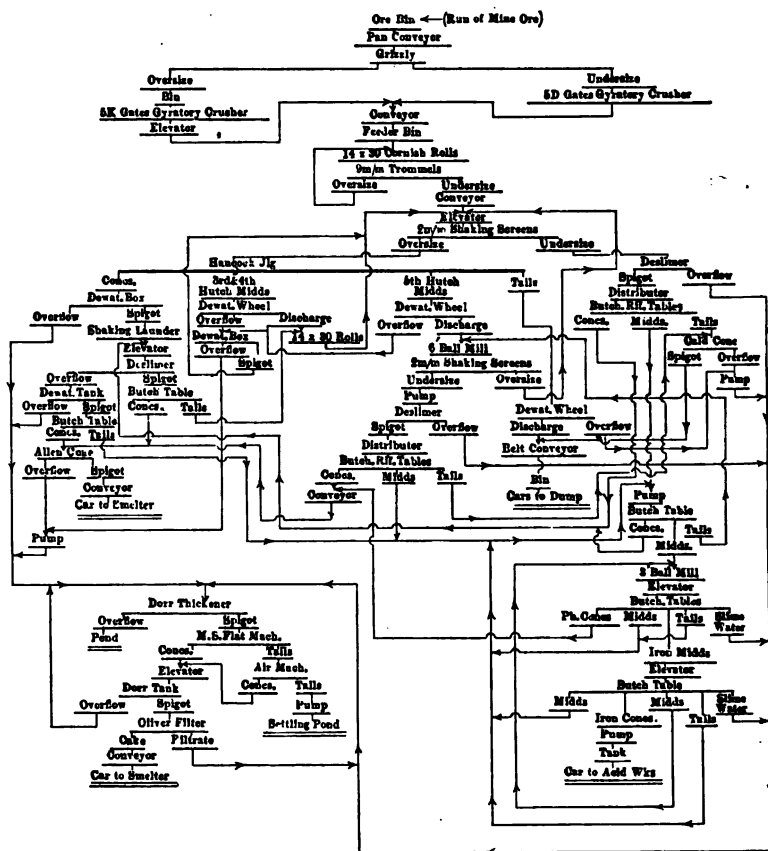


Fig. 1.

In 1909, the Doe Run Lead Co., now a subsidiary of the St. Joseph Lead Co., built a 2400-ton mill at Rivermines, Mo., which in 1913 was enlarged to handle 3600 tons per day. This plant received ore crushed to 4 in. at the mines and, after reducing to 9 mm. at the mill, concentrated it on Hancock jigs and Wilfley tables.

In 1912, the Bonne Terre mill was partially remodeled, Hancock jigs, Richards classifiers, and automatic disposal of mineral and chat were





one No. 5 style K Gates gyratory crushers. A screen analysis of run-of-mine ore feeding the crushers is shown in Table 1.

TABLE 1.—*Screen Analysis of Ore at the Bonne Terre Mill*

| Size                | Per Cent. Weight |
|---------------------|------------------|
| On 10 mesh.....     | 11.1             |
| On 8 mesh.....      | 12.6             |
| On 6 mesh.....      | 7.6              |
| On 4 mesh.....      | 32.5             |
| Through 4 mesh..... | 36.2             |
| Total.....          | 100.0            |

Grizzlies with bars spaced with  $7\frac{1}{2}$ -in. clearance are set on an incline over the two No. 5 D crushers that crush the undersize. The oversize gravitates to a small bin and is crushed by the No. 5 K crusher, and is then elevated by bucket elevator to pan conveyors, which convey all the crushed discharge to belt conveyors. These belt conveyors distribute the ore to feeder bins, scrapers being used to rake it off into each bin. Messiter electric scales attached to the pan conveyors receiving the discharge from the crushers were used for a time, but have been discontinued because of difficulty in keeping them accurately regulated. Table 2 shows a screen analysis of the crusher discharge.

TABLE 2.—*Screen Analysis of Crusher Discharge*

| Opening,<br>Diameter | Weight,<br>Per Cent. | Opening,<br>Diameter | Weight,<br>Per Cent. |
|----------------------|----------------------|----------------------|----------------------|
| + 3 in.              | 12.9                 | + 1 in.              | 14.9                 |
| + $2\frac{1}{2}$ in. | 11.4                 | + 9 mm.              | 12.5                 |
| + 2 in.              | 17.6                 | + 2 mm.              | 9.9                  |
| + $1\frac{1}{2}$ in. | 15.7                 | + 150 mesh           | 3.5                  |
|                      |                      | - 150 mesh           | 1.6                  |
|                      |                      | Total.....           | 100.0                |

The same Cornish rolls that were installed in the mill when it was built in 1883 are still in use. Thirteen sets are used, fed by Challenge feeders. The shells are 14 by 30 in., one chilled cast iron and the other chilled steel, being used on each set. A groove  $\frac{3}{4}$  in. deep and  $1\frac{1}{2}$  in. wide is cut across the face of each steel roll shell to prevent slipping of the ore and consequent grooving of the shell. The roll speed is 10 r.p.m. Weight boxes are used on the rolls instead of springs. The roll discharge is screened through a 9-mm. trommel, 3 ft. (0.91 m.) diameter by 8 ft. (2.5 m.) long, with a slope of 1 in. (25.4 mm.) per foot. The trommel revolves at a speed of 36 r.p.m. The oversize is returned by a bucket

elevator to each roll. The rolls average 185 tons per 24 hr., and have shown by test that 250 tons per hour can be maintained. The rolls are driven by two 75-hp. and one 100-hp. motors with a total load of 210 hp., or an average of 16 hp. per roll, including the oversize return elevator.

### *The Concentration System*

The mill is divided into four sections after crushing the ore, an equal tonnage being treated in each section. The mill building is built on a level site and it is necessary to elevate and pump the various products for treatment. All line shafts are in the basement with the rolls and other heavy machinery on concrete foundations near the floor level. The undersize from the 9-mm. trommels is conveyed to elevators and water added to sluice the ore into the boots. The elevators are 53-ft. (16-m.) centers, using 8 by 16-in. (20 by 40-cm.) buckets with a belt speed of 400 ft. (121.9 m.) per minute. The discharge from the elevators is screened on 2-mm. screens.

The screens are of the Ferraris shaking type, first used by the Desloge Lead Co. in this district, but have been redesigned at Bonne Terre to withstand the service required and to fit the limited head room. The screens are made at the plant, and cost about \$125 each, erected. Three screens are used at each elevator, driven separately from a line shaft, so that repairs can be made, or screen sheets changed, while the other two carry the load. The screen frame is supported by hickory hangers set at an angle of 65° to the frame. The drive-rod is connected to the head of the screen and a jig eccentric on a countershaft over the screen supports. The eccentric is set for 1¼-in. stroke and a speed of 270 to 290 r.p.m. A clear-water spray is used over the screen near the center to wash the fines from the oversize. Various screen openings have been tried, but at present 2-mm. round holes in No. 18 gage sheets are used, the sheets lasting about 40 days. Each screen treats 250 to 300 tons per 24 hr. and requires 0.75 hp.

### *The Jig Installation*

The Hancock jigs treat the oversize of the 2-mm. screens. They are divided into five hutches and a tail compartment. The first and second hutches produce concentrates, sent to the recleaning system. The third hutch is arranged for sending its product either to the concentrates or the middlings. The fourth and fifth hutches produce middlings. The cam shaft has a speed of 65 r.p.m., giving the jig 195 strokes per minute. The horizontal throw averages ¾ in. (19.05 mm.) and the vertical throw about ⅝ in. (15.875 mm.). The tailings are dewatered by drag scrapers in the last compartment in order to separate them from the slimy water



that overflows the jig and goes to the Dorr thickeners. The jig tailings are then sluiced to concrete tanks and dewatered by shovel wheels, the discharge going to a belt conveyor and concrete bin. These shovel wheels are 14 ft. (4.3 m.) in diameter, and make 1 r.p.m., the discharge averaging 15 per cent. moisture.

The third and fourth hutch middlings are dewatered by a shovel wheel and recrushed by rolls, returning to the jig elevator and 2-mm. screens. These middlings are crushed to an average of 3 mm., releasing the galena in as coarse a size as possible. The fifth hutch middlings, averaging from 1.5 to 3.0 per cent. Pb, are dewatered by a shovel wheel and sent to a 6-ft. ball mill, of which there are two types in the plant.

### *The Ball Mills in Use*

A 6-ft. by 22-in. Hardinge mill crushes the middlings in section 1, and a 6 by 4 ft. Allis-Chalmers granulator is installed in section 3. They are being tested for results before placing mills in the other two sections. As final results had not been worked out on either type of mill at the time of writing, it is possible to give only some of the experimental data at present, showing the grinding efficiency and tonnages treated by each mill.

Results with the Hardinge mill are shown in Table 3.

TABLE 3.—*Data on Hardinge Mill Operation*

|                          |   |
|--------------------------|---|
| Average tonnage ground.. | 250 tons per 24 hr. (this includes 50 tons of middlings returned from table treatment). |
| Ball load.....           | 14,000 lb.  |
| Kind of balls.....       | Cast iron, with 3 in. maximum diameter.   |
| Power required.....      | 53 hp.  |
| Speed of mill.....       | 30 r.p.m.   |
| Moisture .....           | 50 per cent.  |
| Ball consumption.....    | 0.9 lb. per ton crushed.  |

The screen analysis in Table 4 shows the crushing performed on this class of material.

The discharge from the mill is screened through a 2-mm. shaking screen. The oversize returns to the jig elevator as jig feed, and the undersize is pumped to a deslimer for table and flotation treatment.

Various tests have been tried in crushing by the Hardinge mill, *e.g.*, crushing the entire middlings, returning the oversize to the jig and the undersize to tables and flotation. This method enriched the flotation feed too much, due to the rich middlings from the third and fourth hutches of the jig. Another test was to crush the entire tailings of the jig, the middlings being crushed in the rolls and returning to the elevator, and the oversize of tailings from the ball mill being discarded as chat. The

ball mill lacked capacity to treat the necessary tonnage in this manner, and it was found that the most economical method was to crush the middlings from the fifth hutch of the jig. This gave low tailings on the jig and low-grade slime for flotation, besides allowing a larger tonnage of ore to be treated in the mill section.

TABLE 4.—*Screen Analysis of Hardinge Mill Material*

| Feed        |                      |                                     | Product     |                      |                                     |
|-------------|----------------------|-------------------------------------|-------------|----------------------|-------------------------------------|
| Mesh        | Per Cent.,<br>Weight | Cumulative,<br>Per Cent.,<br>Weight | Mesh        | Per Cent.,<br>Weight | Cumulative,<br>Per Cent.,<br>Weight |
| On 3        | 5.34                 | 5.34                                | On 4        | 1.1                  | 1.1                                 |
| On 4        | 15.00                | 20.34                               | On 6        | 1.3                  | 2.4                                 |
| On 6        | 12.45                | 32.79                               | On 8        | 2.0                  | 4.4                                 |
| On 8        | 17.45                | 50.24                               | On 10       | 5.8                  | 10.2                                |
| On 10       | 15.60                | 65.84                               | On 14       | 6.4                  | 16.6                                |
| On 14       | 10.27                | 76.11                               | On 20       | 6.6                  | 23.2                                |
| On 20       | 4.49                 | 80.60                               | On 28       | 7.8                  | 31.0                                |
| On 28       | 2.29                 | 82.89                               | On 35       | 6.9                  | 37.9                                |
| On 35       | 1.52                 | 84.41                               | On 48       | 6.6                  | 44.5                                |
| On 48       | 1.95                 | 86.36                               | On 65       | 5.8                  | 50.3                                |
| On 65       | 2.46                 | 88.82                               | On 100      | 7.8                  | 58.1                                |
| On 100      | 3.98                 | 92.80                               | On 150      | 8.4                  | 66.5                                |
| On 150      | 5.25                 | 98.05                               | On 200      | 8.8                  | 75.3                                |
| On 200      | 1.10                 | 99.15                               | Through 200 | 24.7                 | 24.7                                |
| Through 200 | 0.85                 | 0.85                                |             |                      |                                     |
| Total.....  | 100.00               |                                     |             | 100.00               |                                     |

### *The Allis-Chalmers Granulator*

This mill has a grate through which the crushed product must pass before being discharged. As with the Hardinge mill, various tests are being made to determine its crushing capacity, efficiency, etc. The first tests were to crush the jig tails, cast-iron balls with 3-in. (76.2 mm.) maximum diameter being used as grinding mediums. Ball loads ranged from 9000 lb. to 14,000 lb. (4082 to 6350 kg.), but the results were not satisfactory, the tonnage building up in the mill and choking it. Raising the pulp level improved the grinding slightly. It was found, however, that the balls broke up badly, so a change was made to forged chrome-steel balls of larger diameter. Tests for ball charges, load and pulp level have resulted in gradually raising the tonnage of feed and grinding efficiency. The results shown in Table 5 are the best that have been obtained as yet, further tests being made with a closed circuit.

TABLE 5.—*Allis-Chalmers Ball Granulator Results*

|   |   |
|---|---|
| Character of feed.....                      | Jig tails and fifth-hutch middlings.  |
| Ball load.....                              | 12,000 lb. (consisting of 1000 lb. of 5 in. diam., 3000 lb. of 4 in. diam., 8000 lb. of 3 in. diam.). |
| Revolutions per minute.....                 | 23  |
| Grate opening.....                          | $\frac{3}{16}$ in.  |
| Pulp-level distance from center             | 16 in.  |
| Tons feed per 24 hr.....                    | 486   |
| Per cent. moisture.....                     | 62.6  |
| Horsepower.....                             | 59.4  |
| Tons crushed through 2 mm.                  |   |
| per horsepower.....                         | 6.7   |
| Oversize of 3-mm. screen discarded as chat. |   |

| <i>Screen Analysis of Product</i> |                      |
|-----------------------------------|----------------------|
| Mesh                              | Per Cent.,<br>Weight |
| On 10.....                        | 18.7                 |
| On 150.....                       | 62.7                 |
| Through 150.....                  | 18.6                 |
| Total .....                       | 100.0                |

The discharge from the mill for a while was screened through a 2-mm. screen, the oversize being discarded as chat. Later a 3-mm. screen was substituted, resulting in the table tailings averaging practically the same as with the smaller opening. The oversize of this screen was also discarded, but is now being returned to the jig feed, making a closed circuit on the ball mill. Results from this at present are not complete.

#### *Summary of Ball Mill Practice*

It is impossible now to give final results metallurgically as to the effect of ball mills on the milling practice of the ores. However, an increased capacity for the plant is obtained when the middlings are taken from their former closed circuit with the jig and recrushed in ball mills. The undersize of the ball-mill discharge is deslimed, the sands going to separate tables. This gives the jig capacity for more original ore, and the plant now handles 2300 to 2400 tons per day. This alone has justified the installation of the ball mill. On the other hand, rolls cannot grind the low-grade middling as well as the ball mill, the latter giving more material for tables and flotation, which result in lower tailings than the jigs. The regrinding of jig tailings has resulted in lowering the final tailings. Approximately 60 per cent. of the feed to the Allis-Chalmers ball mill is sent to tables, the tailings of which average less than 0.4 per cent. Pb. The flotation plant receives 18 per cent. of the feed and will make minimum tailings there. The tests have not been completed to show the economic effect of lowering the mill tailings, but the results point toward a profit, even with a low-price lead.

### Table Concentration

In the summer of 1915, it was decided by the management to give the Butchart riffing a trial in table practice, in order to increase the tonnage and efficiency of the mills. A series of tests was carried out for several months at the Bonne Terre mill to determine the best application of the riffle to the treatment of these ores.

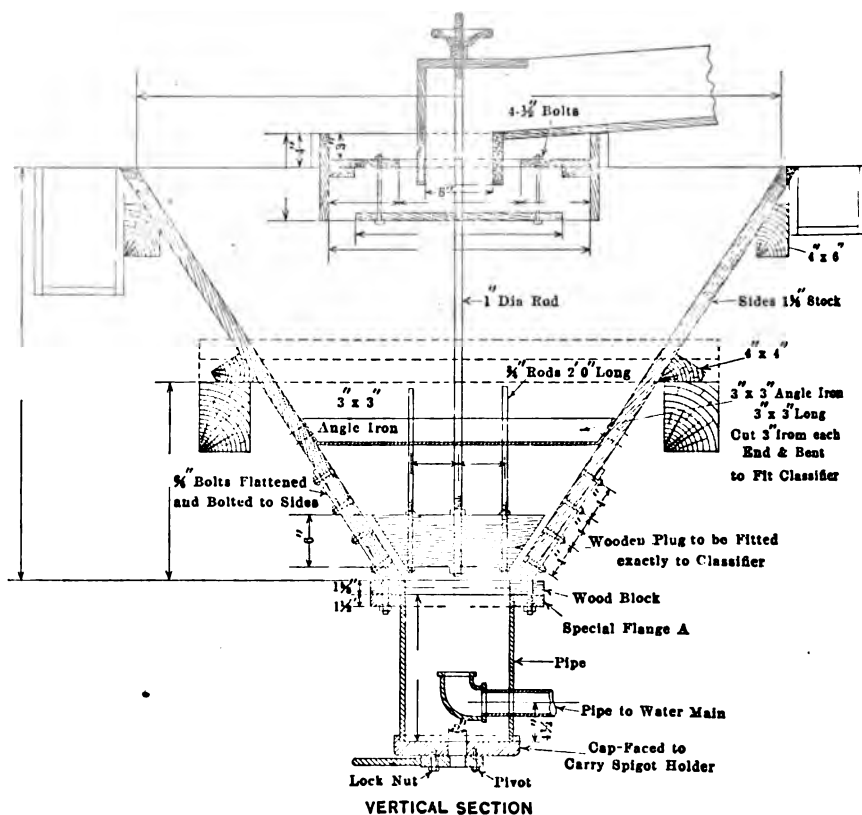


FIG. 4.—THE DELANO DESLIMER.

Previous to this time, Wilfley tables and riffing had been used, the feed to the tables being the spigot products of Richards four-spigot, vortex classifiers. The Butchart riffing worked best, however, with deslimed sands, no separation into different sizes being required. As it became necessary to mix the spigot products of the classifiers, it was decided to change the classification system.

Experiments were conducted to produce a one-spigot classifier that would handle the entire tonnage of the undersize from the 2-mm. shaking screens and deslime it so as to give a table feed practically free from the

minus 150-mesh slimes. The experiments resulted in a one-spigot, two-product, hydraulic classifier, or deslimmer, with an inverted conical or pyramidal tank, the entire perimeter being used for the overflow into a surrounding launder at the upper edge. The deslimmer is provided with a central feed box, having a baffled bottom to break the velocity of the inflowing pulp. The bottom, or discharge chamber, has a clear-water inlet, which directs the flow upward along the axis of the deslimmer. The spigot is inserted in a movable holder, attached to the bottom of the discharge chamber, permitting an instant change in the diameter of opening without wetting the operator. Between the feed box and discharge chamber is a valve with sides parallel to the tank and suspended on the vertical center, and which can be raised or lowered by means of a hand wheel on the suspension rod. This produces a narrow channel between the valve and tank through which the hydraulic water rises and the sands fall.

The feed entering the deslimmer is checked by the baffle, causing the sands to settle through a narrow, rising current of water surrounding the valve. This current increases in velocity until the discharge chamber is reached, by which time the sands are thoroughly deslimed. A hydraulic pressure of 5 or 6 lb. per square inch is maintained in the discharge chamber. The width of the sorting column may be varied by raising or lowering the valve. No whirling or boiling currents are possible and the steady rising current gives a very close separation. The great efficiency of the deslimmer is due to the better application and direction of hydraulic water supplied. A deslimmer 5 ft. 6 in. in diameter treats from 350 to 400 tons per 24 hr., using approximately 120 gal. of hydraulic water per minute.

TABLE 6.—*Screen Analyses of Deslimmer Material*

| Feed              |                  | Spigot,<br>Per Cent.<br>Weight | Overflow,<br>Per Cent.<br>Weight |
|-------------------|------------------|--------------------------------|----------------------------------|
| Mesh              | Per Cent. Weight |                                |                                  |
| On 100 .....      | 64.70            | 86.8                           | 0.69                             |
| On 150 .....      | 7.41             | 8.2                            | 5.16                             |
| On 200 .....      | 4.10             | 2.3                            | 9.32                             |
| Through 200 ..... | 23.79            | 2.7                            | 84.83                            |
| Total.....        | 100.00           | 100.00                         | 100.00                           |

Experiments with the Butchart riffing were carried out with the following objects in view, viz., to determine size and character of feed, to increase the table efficiency, increase its capacity, simplify the flow sheet, reduce water consumption, power and labor. Its ability to treat unclassified material gave it advantages over the Wilfley riffing by eliminat-

ing the multiple-spigot classifier. The concentrates are discharged at the upper edge of the table, the coarser particles being in the upper riffles instead of mixing with the middlings as in the Wilfley riffling. This gives a larger capacity for both the concentrates and middlings separation and enables much larger tonnages to be treated per table.

Tests were made with feed ranging from minus 9 mm., or jig feed, to the finest sands. In each case the riffling was made especially for the size of feed treated. The separation of free galena as concentrates in each case was satisfactory, but in the coarser sizes the separation of the middlings from the gangue was imperfect and would have necessitated recrushing the entire tailings for retreatment. Final tests showed that the jigs gave a better separation of tailings and middlings on material from 9 to 2 mm., while the undersize of 2-mm. screens deslimed to remove the minus 150-mesh slimes, and treated on tables, gave satisfactory results.

With this size of feed the tables now treat from 50 to 60 tons per day, giving clean concentrates and as low tailings as with the Wilfley when treating 20 to 25 tons per day. In operation the table requires very little adjustment, even with the load varying 5 or 10 tons. The riffling used in this work tapers from  $\frac{7}{16}$  in. (11.1 mm.) to  $\frac{1}{8}$  in. (3.2 mm.), with a slope of 1 in 4 in the cleaning zone. The middlings average approximately 10 per cent. of the feed to the tables. This is retreated on a separate table for two mill sections, the tailings being reground in the 6-ft. ball mill. A concentrate cut is made from this table and the middlings are reground in a 3-ft. Hardinge mill and sent to another circuit. In changing the mill from Wilfley to Butchart system, one Butchart table replaced three Wilfleys, and enabled a saving in power, labor and water. Various pumps, launders, classifiers and settling tanks were cut out, making a much simpler flow sheet.

The possibility of using the Butchart riffling to raise the grade of the mill concentrates was suggested by its large capacity and its ability to treat coarse material, discharging the coarsest particles at the upper edge of the table. The jig concentrates were erratic, ranging from 60 to 70 per cent. Pb and occasionally going lower. Before this they had been washed in a log washer or trunking machine, which made a rather poor separation. Tests with the Butchart riffling showed that the feed should be deslimed and treated on two tables. The spigot of the deslimer or plus 100-mesh material was treated on one table, and the overflow, or minus 100-mesh material, was settled in a tank, the spigot feeding the second table.

Table concentrates were mixed with the jig concentrates to produce a mixture of sizes that would bed well on the table. The riffling on each table was especially arranged for the size and character of the feed. The coarse galena was discharged in the upper riffles, followed by the finer

sizes. A good separation was made in the cleaning zone, the tailings constituting true middlings containing practically no free galena. They averaged 25 to 35 per cent., and are recrushed in the jig-middling rolls, returning to the circuit by the jig elevator. The coarse table now treats from 70 to 100 tons per day and has capacity for a larger range in tonnage.

The fine-table concentrates averaged 80 per cent. Pb, 95 per cent. passing a 200-mesh screen. The tailings average 6 per cent. Pb, and are returned to the sand middlings table.

At present the entire jig concentrates and a portion of those from the tables are treated in the recleaning system. The grade of the mill concentrates has been raised from an average of 68 per cent. to 73.5 per cent. resulting in smaller freight and smelter treatment charges, and without affecting the mill recovery.

### *Concentrates Disposal*

The concentrates from the entire mill are sent to a 6-ft. Allen cone for dewatering. The spigot discharges on a conveyor belt to the car, while the overflow returns to the mineral elevator.

The jig-tailings disposal was referred to in describing the jiggling. The tailings from the tables are dewatered by Caldecott cones, the sands discharging through the spigot to the chat conveyor. The cones will average from 400 to 500 tons of minus 2-mm. sands per day, the average moisture in spigot being 28 per cent. A screen analysis of the spigot discharge is shown in Table 7.

TABLE 7.—*Caldecott Cone Spigot Discharge*

| Mesh      | Per Cent.,<br>Weight | Mesh       | Per Cent.,<br>Weight |
|-----------|----------------------|------------|----------------------|
| + 10..... | 2.59                 | + 65.....  | 7.27                 |
| + 14..... | 9.61                 | + 100..... | 6.65                 |
| + 20..... | 17.64                | + 150..... | 5.18                 |
| + 28..... | 22.95                | + 200..... | 1.25                 |
| + 35..... | 15.13                | — 200..... | 0.86                 |
| + 48..... | 10.87                |            |                      |
|           |                      | Total..... | 100.00               |

The sands from these cones, and the shovel-wheel discharge, or jig tailings, are conveyed by a 20-in. (0.5-m.) conveyor to a concrete chat bin with a capacity of 250 tons. Railroad cars of the A-dump type run under this bin, and are loaded by slide gates in the bottom of the bin, and sent to the chat dump. A portion is sold for railroad ballasting and concrete work.

### *The Pyrites Concentrates*

The ore assays approximately 5 per cent. Fe, consisting mainly of pyrite, marcasite and chalcopyrite. This is concentrated in the table

middlings and requires fine grinding to separate the galena, pyrite and dolomite. The middlings from the sand tables are reconcentrated on a middling table and the middlings from this table are sent to a 3-ft. Hardinge mill for regrinding. The mill is operated with a 1000-lb. ball load, replenished with 2-in. cast-iron balls. The product will nearly all pass a 65-mesh screen, and is elevated to a Butchart table. Lead concentrates, assaying 80 per cent., are sent to the Allen cone and middlings are retreated on another Butchart table, which produces galena-pyrites concentrates. The analysis of concentrates shows an average of 41 per cent. sulphur, a portion being shipped to a sulphuric-acid plant for experimental purposes, with a view to recovering the sulphur as acid. The remainder is mixed with the galena concentrates and is shipped to the St. Joseph Lead Co.'s smelter at Herculaneum for its value as an iron flux.

### *Treatment of Slimes*

The overflow of the deslimers, chat-wheel pits, Caldecott cones and various small settling boxes, is thickened in five Dorr tanks, four of which are 40 by 8 ft. (12 by 2.5 m.), and one 50 by 6 ft. (15 by 1.8 m.). The feed to these tanks averages 2.5 per cent. solids, the total flow being approximately 3500 gal. per minute. The solids average 540 tons per 24 hr., or approximately 12 sq. ft. (1 sq. m.) of settling area per ton of dry slime. The central shafts revolve once in 10 min. The overflow from the tanks returns to the mill pond for use as mill wash water, while the spigot discharge is sent to the flotation machine.

The thickened pulp at a ratio of 2.5 to 3.0:1 is treated in a 23-cell Minerals Separation flotation machine, with 24-in. agitating compartments. Creosote oil (hardwood) is added in the feed box and at several alternate cells, a small oil pump being used to give a regular oil feed. The impellers are belt driven from a line shaft, the speed being 335 r.p.m.

The machine is driven by a 100-hp. motor using 83 hp., or an average of 3.8 hp. per cell. The tailings from this machine are retreated in a pneumatic machine. The froth, or concentrates, averaging 50 per cent. Pb from both machines, are elevated by a bucket elevator to a Dorr thickener.

Oil consumption averages 0.55 lb. per ton of dry slimes treated. The slimes average 540 tons per day, or, at present, 23 per cent. of the ore milled. This percentage is higher than in the other mills of the company, due principally to the use of Cornish rolls and the larger proportion of middlings reground in ball mills.

### *Concentrates Disposal*

The froth is broken up by clear-water sprays and thickened in a 38 by 6-ft. (11.6 by 1.8-m.) Dorr tank. The overflow goes to the slimes



tanks and the spigot discharge, averaging 60 per cent. solids, is sent to a 12-ft. by 11-ft. 6-in. (3.7 by 3.5-m.) Oliver filter. The Dorr-thickener feed averages 90 per cent. moisture. A permanent froth remains on the surface and is kept from overflowing by a baffle ring 12 in. (0.3 m.) inside the overflow and 12 in. above and below the surface of the water. The central shaft revolves twice per hour.

The cake from the Oliver filter averages 15 per cent. moisture, and is conveyed by belt conveyor to cars and shipped to the smelter. At present a portion of this cake is dried in vats, fitted with steam coils, to a moisture of 7 per cent., in order to enable the smelter to handle the output. The filtrate returns to the slimes thickeners. The filter speed is one revolution in 6 min. and 30 sec. The wet cake averages approximately 50 tons per 24 hr. One "blow" compartment is maintained about 6 in. above the scraper, the other compartment blowing after immersion in the pulp, which is kept at a maximum level. The blow pressure ranges from 5 to 10 lb. The vacuum is produced by a No. 5 Root pump, averaging 25 to 20 in. (0.635 to 0.508 m.). The vacuum requires 29.5 hp. and the filter 1.7 hp. The life of a canvas and burlap averages 3 months. After a new canvas is in use from 4 to 6 weeks, steam is exhausted in the pulp for the purpose of heating. This gives a larger capacity without increasing the moisture in the cake. Acid washes have been tried for the canvas without success.

The tailings from the flotation machines are pumped by a 3-in. Morris sand-dredge pump to the impounding pond, against a total head of 50 ft. (15 m.) using 18 hp. The pond dam is built of chat hauled from the mill and is 50 ft. high at its maximum point. The flow of pulp averages 225 gal. (851 l.) per minute, and immediately spreads over the surface of the pond, allowing the solids to settle rapidly. No water remains on the surface, as it filters



FIG. 5.—DOE RUN LEAD CO'S MILL, RIVERMINES, MO.

rapidly through the chat dam. Consequently the pond will eventually fill up in layers, forming a smooth dry body of compact slime. The filtered water is perfectly clear and free from slime.

The mill water is returned to a pond with an area of 17,000 sq. ft. (1579.3 sq. m.), formerly used for settling the mill slimes before the installation of Dorr tanks. Any slimes now escaping the Dorr tanks are caught in this pond and pumped back periodically for retreatment. The volume of water used in the mill circuit is approximately 3500 gal. (13,248 l.) per minute, or a ratio of 9:1. This compares with a ratio of 17:1 before the change was made in table practice, classification, etc. Loss in mill water, due to pumping to waste of flotation tailings, etc., is made up of mine water and from a pumping station on Big River, 3 miles away.

All machinery in the mill is electrically driven, using an average of 7.29 kw. per ton of ore treated. With the present tonnage treated this equals approximately 925 hp. As much machinery as possible is driven by individual motors, even some of the tables using separate 1-hp. motors. All motors are loaded to capacity to obtain greatest efficiency.

The labor in the mill is all American and much more efficient than a few years ago. The tons of ore milled for 1916 averaged 25 per man shift, with an average of 84 men employed per day, including maintenance. Three shifts are operated per day, 6 days a week, Sundays being used for repairs. The mill recovery varies with the grade of ore, the tailings from the plant being equal to the district average. Future improvements will probably include larger crushers, substitution of other grinding machinery for the Cornish rolls, and extension of table and flotation equipment, with the view of raising the tonnage, recovery, and the cheapening of costs.

### THE RIVERMINES MILL

The Rivermines mill is divided into six sections. The ore is crushed to 4 in. at the mine shafts and hauled in A-dump cars by railroad to a large storage bin extending across the mill end. This bin has a capacity of 3500 tons or sufficient for nearly a day's run. A trestle extends beyond the bin, so that the cars may be pinched down grade over the bin and emptied. In winter, steam nozzles are used to thaw the ore sufficiently for unloading. All the ore is weighed on scales on this trestle.

The ore is fed from the bin by roller feeders, operated by a gripping cam, to 1-in. (25.4-mm.) round-hole trommels, 8 ft. (2.5 m.) long by 24 in. (0.6 m.) in diameter. The oversize of these trommels is crushed by two No. 3 McCully gyratory crushers, the crusher discharge joining the undersize from the trommels, and is carried by belt conveyor to two 48-in. by 12-ft. (1.2 by 3.7-m.) trommels with 9-mm. round holes. The oversize from these trommels is recrushed by rolls. The discharge from the rolls

returns to the conveyor feeding the 9-mm. trommels, the undersize of which goes to an elevator, which raises the dry ore to the 2-mm. trommels on the top floor. From this point the milling practice is practically the same as at the Bonne Terre mill.

### *The Marcy Mill*

An 8 by 6-ft. (2.5 by 1.8-m.) Marcy ball mill has been installed in section 5 for experimental purposes. This mill has been tried in various tests and is now crushing the oversize from the 9-mm. trommels and the dewatered middlings from the fourth and fifth hutches of the Hancock jig. The discharge from the ball mill goes to the 2-mm. trommels, the oversize feeding the jig and the undersize the tables and flotation plant. As in the Bonne Terre mill, final results have not been worked out on this mill. As it is a grate-type mill, the results are similar to those of the Allis-Chalmers granulator. In this case 73 per cent. of the feed to the mill is the oversize of the 9-mm. trommels, the balance being middlings from the jigs and tables, the total averaging 750 tons per day. Approximately 200 hp. is required, the mill using a ball load of 28,000 lb. (13,800 kg.) of 5-, 4- and 3-in. balls; 5-in. balls being used for replenishing. The average moisture is 65 per cent. Screen analyses of the feed and discharge product are shown in the table.

### *Marcy Mill Results*

| Feed       |                   |                              |                            | Discharge  |                   |                              |                             |
|------------|-------------------|------------------------------|----------------------------|------------|-------------------|------------------------------|-----------------------------|
| Mesh On    | Per Cent., Weight | Cumulative Per Cent., Weight | Per Cent. Total Pb Content | Mesh On    | Per Cent., Weight | Cumulative Per Cent., Weight | Per Cent., Total Pb Content |
| 1¼ in.     | 27.7              | 27.7                         | 27.1                       | 3 mm.      | 20.1              | 20.1                         | 5.2                         |
| 9 mm.      | 41.3              | 69.0                         | 44.5                       | 2 mm.      | 9.1               | 29.2                         | 3.6                         |
| 3 mm.      | 14.1              | 83.1                         | 16.5                       | 20         | 21.9              | 51.1                         | 10.0                        |
| 2 mm.      | 7.8               | 90.9                         | 3.4                        | 28         | 8.4               | 59.5                         | 6.8                         |
| 20         | 4.4               | 95.3                         | 1.5                        | 35         | 5.7               | 65.2                         | 5.5                         |
| 35         | 2.0               | 97.3                         | 1.5                        | 48         | 5.2               | 70.4                         | 7.7                         |
| 65         | 0.9               | 98.2                         | 1.2                        | 65         | 4.0               | 74.4                         | 6.8                         |
| 150        | 0.8               | 99.0                         | 1.2                        | 100        | 3.9               | 78.3                         | 7.7                         |
| -150       | 1.0               | 1.0                          | 3.1                        | 150        | 4.0               | 82.3                         | 9.0                         |
|            |                   |                              |                            | 200        | 3.0               | 85.3                         | 6.1                         |
| Total..... | 100.0             | ....                         | 100.0                      | -200       | 14.7              | 14.7                         | 31.6                        |
|            |                   |                              |                            | Total..... | 100.0             | ....                         | 100.0                       |

The jig tailings are dewatered in chat wheels as at Bonne Terre, and the table tailings in sand cones and a sand drag. They are then stacked

by belt conveyors, forming a very large tailings pile, which has accumulated since the plant started.

The tailings from the flotation plant are pumped by a 5-in. Morris sand-dredge pump to the top of the chat pile and are used for spreading the chat from the conveyors.

The mill water is settled in a pond back of the plant and is pumped from there to a reservoir situated at a maximum elevation in front of the building, from which it is re-used in the mill circuits. The volume of water used is approximately 5000 gal. per minute, or a ratio of 7:1 of ore.

All power is alternating current, an average of 8.24 kw. is used per ton of ore treated. The labor is all American. This mill will be rear-



FIG. 6.—FLOTATION MACHINE, RIVERMINES MILL.

ranged later with ball mills for reducing the general tailings and increasing the capacity.

#### THE LEADWOOD MILL

The Leadwood mill has practically the same milling operations as at the Bonne Terre and Rivermines plants. All the ore is hoisted from No. 12 mine shaft at the mill and is conveyed direct to the crushers. In case the hoist should be out of commission for any reason, the ore from the outlying shafts can be diverted to an ore bin from which it is conveyed to the 5 K Gates crusher, thus preventing an entire shutdown of the mill.

The regular ore feed is screened on a grizzly with bars spaced 8 in. apart. The undersize is crushed in the 5 K crusher and the oversize in the 6 K crusher. The discharge from these crushers is screened through 1½-in. trommels, the oversize being recrushed in three No. 5 D Gates

crushers. From here it is conveyed to an ore bin of 1600 tons capacity, and distributed by a tripper on the conveyor.

The ore from the bin is screened through 9-mm. trommels. The over-size is recrushed by rolls and returned to the 9-mm. screens. The under-size is screened through  $1\frac{1}{2}$ -mm. trommels. From this point the concentration is practically the same as at Bonne Terre and Rivermines.



FIG. 7.—THE BONNE TERRE MILL, ST. JOSEPH LEAD CO.



FIG. 8.—ST. JOSEPH LEAD CO'S MILL, LEADWOOD, MO.

The jig tailings discharge in the bottom of the tail hutch of the jig and fall over stationary  $1\frac{1}{2}$ -mm. screens. The oversize discharges on the chat conveyor, while the water drains to the sand drag. Wilfley tables, a portion Butchart riffled, are used for sand concentration. The minus  $1\frac{1}{2}$ -mm. and plus 150-mesh sands are treated on the coarse tables, the tailings going to a concentrates pit. A drag scraper in this pit de-

waters the sands and discharges them onto the chat belt with 15 percent moisture. This drag gives some trouble by breaking, but manganese-steel link chains are now being used in order to reduce this to a minimum.

The fine sands, minus 150-mesh, and slimes are thickened in settlers and concentrated on Wilfley tables, the tailings going to the Dorr tanks at the flotation plant. The overflow of the sand drag also goes to the flotation plant. The slimes are thickened in four 36 by 8-ft. Dorr thickeners. The pulp is agitated in a 16-cell Minerals Separation flotation machine, the same oil being used as at the other mills. The tailings are retreated by an air machine, and the concentrates are thickened and filtered by an Oliver filter. The flotation plant is lightly loaded, approximately 350 tons dry slimes, and the concentrates produced average 58 per cent. Pb. The slime tailings are pumped to a settling pond behind the chat dump and settle as at Bonne Terre.

The mill chat, jig and table tailings, are conveyed by belt conveyor to a tailings pile. This conveyor inclines at an angle of  $26^{\circ}$  and the chat is sufficiently dry not to cause any slippage on the belt, which is not cupped.

This mill will be equipped with ball mills for regrinding middlings and tailings as at Bonne Terre. A 6 by 4-ft. Allis-Chalmers ball granulator is installed in one section, but has not started operating, so no data can be given. A 3-ft. Hardinge mill is installed for regrinding table middlings as at Bonne Terre, but is just starting operation, and no results are obtainable. The Leadwood ores contain considerable zinc sulphides, which will probably concentrate in this circuit.

The milling practice of the St. Joseph Lead Co. is passing through a transitory stage. Considerable experimental work is being done at each plant, a thoroughly equipped laboratory being maintained at each mill. While large tonnages are being treated in the three plants, with the highest recoveries obtainable, it is probable that the introduction of ball mills and further extension of flotation equipment will greatly change the present practice.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Some Economic Factors in the Production of Electrolytic Zinc

BY R. G. HALL, \* B. S., ST. LOUIS, MO.

(St. Louis Meeting, October, 1917)

AN article on the subject of electrolytic zinc no longer needs to be preceded by an apology. The production of zinc by electrolysis is past the laboratory stage and has become an economic factor of considerable importance in the total production of zinc in the United States.

The various methods of procedure to be followed for the greatest economy of manufacture are fairly well known, consequently a detailed description of any method is no longer necessary or desirable. In this paper I shall endeavor to lay down some of the governing conditions in this industry, the relation of methods to ore deposits, freight rates, and sources of power supply.

### POWER POINTS

Assuming for a moment that, for the present at least, the electrolytic production of zinc will be confined to those localities having developed water power, a glance at these powers so developed in the United States, with their relation to known zinc orebodies of economic size, will not be without interest.

#### *Northwest*

In the Northwest we have water powers, developed and undeveloped, in Washington and Montana. These must, of course, bear a close relation to the rapidly growing zinc production of the Coeur d'Alenes and of the Butte districts. In general, the power of the territory in question is relatively low-priced, at least as compared with the cost of developing power from coal, and the near neighborhood of large bodies of more or less complex zinc ores makes these water powers of more than usual interest.

The same remarks may be applied equally to the developed water powers of the Canadian side of the line in the Northwest, where we have again commercially available complex ore deposits such as are being exploited today at and near Trail.

#### *California*

One of the large complex ore deposits of the far West is to be found in Shasta County in the immediate neighborhood of the territory of the

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\* General Manager, River Smelting & Refining Co.

Northern California Power Co., and developments in this connection are said to be now under way. Again, we have the highly desirable combination of a relatively low-priced power with a good-sized available ore deposit.

### *Utah*

In the Utah territory we have certain developed water powers in connection with such ore deposits as the Tintic and the Park City districts. Evidently the relation between the power and the value of the ore deposits is sufficiently close to justify further development, as we see such already undertaken in that territory. Further to the east we have power development at various points in Colorado in the neighborhood of the large and valuable complex ores of the Lake, Summit and Eagle County districts, as well as those of the San Juan district.

The writer is not sufficiently familiar with the cost of large quantities of electric power in the Colorado district to judge whether future development of electrolytic zinc in Colorado will be permitted in connection with these water powers.

### *Southwest*

In the Southwestern country, namely, Arizona and New Mexico, there is not, so far as I know, sufficient development of low-cost power to make attractive any particular effort to develop electrolytic zinc in that territory.

### *Mississippi Valley*

Passing east from the Rocky Mountain territory, we have no further development of power on a large scale until we reach the Mississippi Valley. Here we have the power plant at Keokuk, Iowa, producing power, which, while perhaps not as low-priced as some of the Northwestern powers or as the older Niagara Falls power, is yet, when considered in connection with its economic position in reference to freight rates, to be considered as a point of considerable advantage.

Further north there was formerly some power available at Sault Ste. Marie, Mich. This, however, is so remote except for Canadian or Northwestern ores that it need hardly be considered in connection with the present inquiry.

The development of power in the southern Mississippi Valley has not yet become of sufficient industrial importance to attract any attention for electrometallurgical work.

Further east we have the power developments at Niagara Falls, but from all accounts these are already taken up to the fullest extent and no power is available, at least none in large quantities for electrometallurgical work.



It is not to be considered that the above survey is in any sense complete, but only as more or less indicative of those powers which may be considered as of importance in connection with developments of electrolytic zinc in the future.

### COAL AND FUEL SUPPLY

The electrolytic refining of copper requires not alone power for the electrolytic tanks, but also requires large quantities of coal for heating the solutions with steam heat or otherwise, as well as coal more or less of special quality for cathode furnaces.

In the electrolysis of zinc solutions we do not have this requirement because, in general, the solutions do not require heating, but on the contrary require occasional cooling, and the temperature for the melting down of zinc is so low as not to require a large amount or any special quality of coal for the cathode furnaces. Consequently, for an electrolytic plant alone, depending on central-station power, coal is relatively of very small importance. This does not mean that coal is not to be used at any point in the process, but only that it is not of any importance in the electrolytic process itself.

### ACIDS AND CHEMICALS

The consumption of acids and chemicals in electrolytic-zinc production is relatively quite small. No matter what kind of ore is used as a supply of zinc, unless it should be an oxidized ore with considerable quantities of soluble carbonates other than zinc, the amount of sulphuric acid lost in a cycle is a very small factor, and in a modern electrolytic plant the use of other chemicals than those for laboratory purposes may be considered as nil.

### FREIGHT RATES

The relation of freight charges to grade of ore is an important one and requires to be considered more in connection with the general ore supply than by itself. It is to be noted, however, that the basing point for lead and zinc, the principal metals with which we are concerned in this inquiry, is Mississippi Valley, consequently the value of the refined product must be considered only in connection with some delivery point having a freight rate basis that can be compared to the rate from East St. Louis.

At the present time, of course, with much of our metals going for export, this factor is considerably complicated, but for the purpose of this paper we shall consider the cost of the metals f.o.b. Mississippi Valley points, or equivalent to East St. Louis basis.

## ORE SUPPLIES

As indicated above, the ore supplies most available for the production of electrolytic zinc are those of the Rocky Mountain and far western territory. We shall consider these further in detail with especial reference to their availability for the process in question, and the various factors in their treatment to bring them into the most desirable shape before the real electrolytic treatment begins.

*Joplin Territory*

To those acquainted with the ores and concentrates produced in the Missouri-Kansas-Oklahoma field, it need not be said that at the present time the electrolytic process for zinc recovery has nothing to offer to this territory. The ores are simple in their character; that is, the number of metals concerned is small. They are easily concentrated as compared with most of our zinc ores of the Rocky Mountain territory, and, best of all, they are easily concentrated to a very high tenor of zinc. The concentrate consisting of 60 per cent. zinc (in the form of sulphide) has only about 10 per cent. of total gangue left in the ore. I do not make any prediction as to the future development of the electrolytic process with its relation to the ores from the Joplin territory, but I can say that today the older process in the field has nothing to fear from the newer.

*Northwestern Field*

The developed orebodies in Montana are somewhat in a class by themselves. While they are complex in the variety of metals present, they do not present the complex concentration problems that are found in some other territories of the Rocky Mountain field. For example, the largest producer in the territory seems to find no difficulty in making a concentrate containing 50 per cent. or more zinc and a recovery which, viewed in the light of our experience in concentrating zinc ores of a few years ago, may be considered very remarkable indeed. The concentrate so produced is high-grade and of a character desirable for the retort process of smelting, and this, in general, may be said of all the concentrates being produced, not alone in the Butte district but in the Coeur d'Alenes as well.

The concentrates of the Butte district, however, for the most part contain, besides zinc, a very valuable constituent in their silver, while the elements of less value are lead and copper. Whether the electrolytic process has sufficient attractions to offer to the producers of these ores is a question that must be answered for each individual case, and I think has been answered successfully and conclusively by one large company for its own work.

The principal points that I wish to emphasize in regard to the use of Butte concentrates are their high tenor in zinc and their considerable values in silver, with copper and lead following after in various proportions. Any method of treatment, therefore, must take account of these lesser values as well as of the major value of the zinc itself.

The whole value of the electrolytic process for the treatment of ores of this character must then be considered, not alone the actual cost of reduction of the zinc itself, but also the relative values of the subsidiary metals recovered by the electrolytic process as compared with the direct process of retort smelting.

The recovery of the subsidiary metals, especially the precious metals, which may be obtained by electrolysis, may well be so much greater than the recoveries by the somewhat wasteful retort method as to show a substantial advantage over the latter in ultimate profits. This remark will apply equally to all ores of the Rocky Mountain country where these ores contain values other than zinc.

In the case of the ores of the Coeur d'Alenes district, so far as my limited acquaintance with that territory goes, we have in general a somewhat different condition. They resemble the Butte ores in that a concentrate of a relatively high grade in zinc can be made; but the silver is usually of a lower tenor and the lead of a somewhat higher. In both cases, however, it is to be noted that the element which presents the most difficulty in the treatment of zinc ores; namely, iron, is to a large degree absent. Consequently, the ores lend themselves readily to concentration, either by the older method of gravity concentration, or by the newer flotation, or a combination of both, and the resultant concentrate is such as to be of value either for the retort process of smelting or for the electrolytic process of zinc extraction.

### *California*

The deposits of complex ores in Shasta County, tributary to the distribution lines of the Northern California Power Co., are of course of very great importance and the developments now taking place promise to form a large factor in the production of electrolytic zinc of the United States. These ores, so far as my acquaintance with them goes, are generally of a somewhat complex character and do not lend themselves readily to a gravity separation of the various minerals. Consequently, they have always presented a more than usually attractive field for the electrometallurgist, and about the first really intelligent effort to produce electrolytic zinc, at least the first with which the writer is acquainted in the United States, was made in connection with the large ore deposits in that territory.

*Utah*

The zinc ores of Utah are of great importance, and it is confidently believed that the application of the electrolytic process for the treatment of these ores will bring Utah up to a point where that State will be a considerable factor in the production of zinc. The ores of the Park City district are somewhat simple in character, in that they are readily concentrated to a fairly high tenor in zinc but are also apt to carry very considerable values in silver. In the Tintic district the ores are of an even more complex nature, so far as the writer is acquainted with them, and they have proved in general somewhat difficult to concentrate, classing in this way more with the Colorado materials to be mentioned later.

*Nevada*

In southwest Nevada there are very considerable bodies of the mixed carbonates of lead and zinc. So far these have been handled by wet concentration with considerable success, and on account of their great value as "mixing" ores in the retort process, it is somewhat questionable whether the electrolytic process would have an application here.

*Colorado*

The deposits in practically all of the districts of Colorado are of an entirely different character from those of the Northwest and, to my mind, must be approached in a different manner for intelligent exploitation. Taking the ores of Lake County or the Leadville district as an example, we have a variety of minerals and a complexity of mineral structure rarely occurring in the Northwestern territory.

The primary zinc mineral, "Black Jack" or Marmatite, is in itself a complex of zinc and iron sulphides, which for the most part contains, when pure, only from 40 to 45 per cent. metallic zinc. And, in addition, we have galena, generally in commercial quantities, often crystallized so closely with the zinc-bearing material as to be impossible of separation without extremely fine crushing. Pyrrhotite and pyrite also are constituents which must be reckoned with in the concentration, as these generally carry some precious-metal values and are often crystallized with the other minerals so closely as to be difficult of mechanical separation.

Any method of concentration, therefore, which will extract all of the values from these ores without attempting to separate one mineral from another will necessarily result in a complex concentrate containing zinc, iron, lead (as sulphides) together with more or less silver and gold, with occasionally some copper.

The present method of concentration of these ores is one that has been

worked out in a most ingenious way by the pioneers of the business and for the most part makes three products:

First in value generally, we have zinc concentrate. Averages are somewhat dangerous, but on the whole I believe I am not far from the truth in saying that the concentrates produced from Colorado ores by these methods will not average much over 40 per cent. and will rarely go above 45 per cent. zinc. In addition, they usually carry more or less lead, the balance for the most part being iron in the form of sulphide, with the rock gangue very low. The silver contents will vary all the way from 1 to 2 oz. up to such a point as to make it a considerable factor in the value of the concentrate. It is quite evident that such a complex as this presents equally complex problems in the subsequent metallurgical treatment.

Next in value in general, we have lead concentrate. This concentrate will necessarily show a considerable tenor in zinc and as the silver shows a preference for the lead the silver contents of the lead concentrate is usually considerably higher than that of the zinc concentrate.

Finally, there is a product usually produced from magnetic machines, which is not in all cases of economic value and cannot always be considered as a concentrate, although the writer has sometimes thrown it out as tailings and afterwards sold it as a concentrate, so it depends somewhat on conditions which it is. This is usually a very high iron, low lead, relatively low zinc, and low silver material. For the most part, in the district under discussion, the silver and lead values are very low and, as indicated above, it is often a question whether it is a concentrate or a tailing.

Lastly, there is a rock or gangue matter which is generally eliminated with difficulty and usually carries with it a considerable percentage of the zinc.

Taking the process as a whole, then, it is seen that there are three products of more or less economic value and one of waste, and each one of these carries off with it something which does not belong there, and occasionally carries it off to such an extent that it also carries off the profit of the mine. It is to the handling of these ores that the writer has devoted most of his attention.

The ores of the San Juan district in Colorado differ from the ores of the Lake County and Summit County districts only very slightly in their complex characteristics, and remarks made concerning the one will apply with very slight modifications to the others. I do not mean to intimate that there are no simple zinc ores in Colorado. I have seen such ores there, but the production has not yet become an appreciable factor in the world's production of zinc.

*Carbonate Ores.*—The remarks made above concerning concentration of complex ores in Colorado do not apply to the carbonate

ores of the Leadville district and some other territories in Colorado. These ores are in a class by themselves and have hitherto been handled only for the recovery of their zinc directly by retort smelting, without preliminary concentration.

A certain quantity is now being converted into a zinc concentrate or oxide, for use in the manufacture of pigment, but so far as I know no real attempt at concentration of these ores has been made for the purpose of beneficiating them for manufacturing spelter; at least none has been undertaken on a commercial scale.

### *Arizona*

Complex ores having their principal values in zinc and minor values in copper, lead and silver, have been developed at various points in Arizona, notably in the Chloride and neighboring districts and certain other points in the south central part of the State.

For the most part, these ores present problems similar to those of Colorado in their mineral complexes, having minerals not easily separated from one another, and, so far as the writer is acquainted with the territory in question, remarks on the subject of the Colorado ores will apply, with but slight modifications, to those found in Arizona.

### *New Mexico*

The various fields in New Mexico may be considered in somewhat the same category, some of the ores being even more difficult of concentration than those of Colorado so far as making a high-grade concentrate of zinc is concerned.

Of this Southwestern territory in general, then, it may be said that the ores are found to contain zinc, copper, silver and lead, mentioned in general in the order of their importance, with iron as an invariable accompaniment and rock minerals which have sometimes proved very difficult to eliminate by methods of gravity concentration.

### REQUIREMENT FOR ELECTROLYSIS OF ZINC

The requirement, and I use it in the singular, for the electrolysis of zinc is zinc sulphate and zinc sulphate only. Most other elements found in the solution are harmful to the electrolysis of zinc sulphate. I have never found any that I could confidently say were beneficial. This requirement seems relatively simple, but when it is understood that this process is to be applied to such a complex of ore minerals as is indicated above in the review of ore deposits, it will be understood that the production of zinc sulphate, pure, of the standard strength, is not the easiest problem in the world. And yet it is only this problem that has stood in

the way, for all time past, of the manufacture of zinc electrolytically. All of the factors necessary to the production of a solid coherent plate of electrolytic zinc were known long ago, but it is only recently that we have been able to produce this pure zinc sulphate on a commercial basis and in large quantities and have been able to obtain the electric current at such a cost as to make the production of electrolytic zinc a commercial possibility.

### *Roasting of Ores*

Inasmuch as sulphide of zinc as it occurs in nature is not in general soluble in dilute sulphuric acid for the direct production of zinc sulphate, it is necessary to roast sulphide ores in order to eliminate the sulphur and convert the zinc to an oxide or sulphate.

The roasting of zinc ore for sulphuric acid leaching presents a very different problem from the roasting of the same ore for retort smelting. In the case of retort smelting it is necessary to have the sulphur eliminated to the greatest possible extent. Sulphur remaining behind either as sulphate or sulphide, beyond a certain amount, is harmful.

For the solution of zinc in sulphuric acid, however, only the sulphur remaining behind as sulphide, and not always all of that, is harmful or causes a loss of zinc. Zinc that has been roasted to sulphate is, of course, not harmful and is even desirable, as the sulphuric acid combined therewith helps to make up for the mechanical and other losses in the subsequent process.

Another difference in the two methods of roasting arises from the fact that in the roasting of ferruginous zinc ore for retort smelting little attention is paid to that complex compound of zinc and iron which is generally formed in a roaster from such ores and is usually referred to as a zinc ferrite.

In roasting Colorado ores for the production of spelter, the writer has found as much as one-third of the total zinc in such ore to be insoluble in a relatively strong solution of sulphuric or hydrochloric acid. It is evident that a material such as this would be fatal if formed in quantity when the ore is to be subjected to leaching with dilute sulphuric acid containing only from 6 to 8 per cent.  $H_2SO_4$ .

Fortunately, however, the conditions which produce this so-called ferrite are not such as need necessarily prevail in the roasting of ores for leaching with sulphuric acid. The production of this material in the roaster seems to be a function of temperature, fineness of ore and of time.

In roasting for the production of spelter the temperature is usually raised at the end of the roast for the purpose of decomposing sulphates and the elimination of the last possible amount of sulphur. As pointed out above, in a roast for leaching with sulphuric acid, the presence of

sulphates is rather an advantage than an objection, and consequently the same high temperature is not necessary.

The other factors mentioned do not necessarily vary much in either case. It is evident, however, that in roasting a complex concentrate such as that from Colorado and the Southwest, containing only about 35 to 40 per cent. zinc, most of the balance of the metal contents being iron, there is great danger that an undue proportion of the zinc will be held back, insoluble; at least, such has been the writer's experience.

In roasting zinc ores for the complete elimination of the sulphur, the loss in roasting of fine material, carried out mechanically, and of zinc, lead and silver actually volatilized, is quite appreciable. The writer had the somewhat doubtful pleasure of smelting several thousand tons of some of the earliest concentrate produced by the flotation method in the Montana territory. These concentrates were roasted in the standard type of Hegeler kiln, and by the time they had dropped seven times from hearth to hearth, and leaked out of the doors and other parts of the kiln not made for ores to leak out of, by no means all of the original concentrates found their way into the retort. Subsequent practice has much improved this, but the roasting of a flotation zinc concentrate in a multiple-hearth kiln is not one conducive of sweetness of disposition in the metallurgist.

#### *Extraction of Zinc with Sulphuric Acid*

It is evident that in the leaching with sulphuric acid, of a calcine, produced as above mentioned, other metals than zinc may go into the solution. Contrary to the general belief, the solution of iron is not a difficult one to deal with, as this element is rather easily removed.

In the case of a concentrate containing copper, a considerable amount of the copper will be dissolved with the zinc. This removes the copper from the place where it ought to stay—namely, with the lead and silver—and takes it where it should not be—with the zinc. Consequently it has to be removed by separate treatment. Other soluble metals and metalloids are dealt with in the familiar standard manner which needs no description here.

The handling of the residue from the sulphuric acid leach is, however, one that is of considerably more interest. The metals economically valuable in it are copper, lead and silver, with probabilities of some gold, and the method of smelting must be adopted to the metal of principal value, whether copper or lead.

If the lead is high, as is frequently the case, the material will go to a lead smelting plant where the copper is of secondary consideration. In many cases, notably in the Southwest, the copper is the metal of principal value, and under these conditions it may pay to send the material to a copper smelter and sacrifice the lead.



The losses in leaching are apt to be slight in the case of lead, silver and gold, but may be considerable in the case of copper, depending upon the subsequent treatment of the zinc sulphate, and the method adopted will be such as to fit the individual case.

### *General Conclusions on Roasting and Leaching Processes*

Considering the ore as produced from the mine and carried straight through to the production of the finished metals, along the above indicated lines, we have:

1. *Concentration*.—It has been shown that the concentration of a zinc ore may produce concentrate of a very low or very high tenor in zinc, depending on the other minerals present and the easy separation of these minerals.

As pointed out above, when the ore is concentrated for the purpose of roasting and leaching with sulphuric acid, it would be desirable that a concentrate should be made as high-grade in zinc as possible and as low-grade in iron as possible. The loss in leaching will be directly proportional to the volume of tailings remaining and also somewhat proportional to the amount of iron in the calcines.

2. *Roasting*.—The remarks made above on the roasting of zinc ores for subsequent production of electrolytic spelter sufficiently cover this side of the problem. In general, for an ore high in zinc and low in iron, there will not be much difficulty in roasting to a point that will make possible an economical extraction of the zinc.

3. *Leaching*.—It will be seen that a calcine containing 68 to 70 per cent. of zinc oxide equivalent to about 55 per cent. metallic zinc would have a residue of about 35 per cent. when this residue contains 10 per cent zinc. Whereas, a calcine containing 40 per cent. metallic zinc would have a residue of very nearly 60 per cent. when the residue contains 10 per cent. zinc.

In the first case, we would have a loss of only about 6 per cent. of our original zinc, while in the second case we have a loss of approximately 15 per cent. of the original zinc, and with such a low-grade concentrate as this the loss is likely to be even higher.

It is evident, therefore, that as the grade of concentrate so roasted and leached is lowered the losses in zinc may rise very rapidly, not alone relatively but absolutely, and this loss will become the greater as the percentage of iron increases in the residue.

4. *Power*.—Again it will be seen that only in those cases where the source of current is relatively near by, or where the concentrate can be made high-grade, will it be permitted to follow this method of handling.

Comparison of the total amount of freight paid on the contents of a 50 per cent. zinc concentrate from Montana, landing 85 per cent. of the

product in New York, is of interest as showing the approximate distribution of the freights paid and the relative amount of freight paid in each case.

|   |        |
|---|--------|
| 2000 lb. ore to Mississippi River.....                    | \$7.00 |
| 5 per cent. moisture.....                                 | 0.35   |
| 850 lb. metal to New York.....                            | 1.44   |
|   | <hr/>  |
|   | \$8.79 |
| 850 lb. metal, Montana to New York.....                   | 4.31   |
|   | <hr/>  |
| Difference in freight paid on 1 ton 50 per cent. ore..... | \$4.48 |

From the difference of approximately \$4.48 per ton of ore shown above, in favor of electrolysis at or near the mine in Montana, must be deducted such local freights as may be paid in case the power is not obtainable at the mine itself. This may decrease the above difference by as much as \$1 per ton.

All of these conditions have to a very desirable extent been fulfilled in the Montana and Idaho field, in that there can be made a high-grade concentrate with a high efficiency of concentration, and that there is power close to the ore supply, so that in no case does transportation become a serious matter. I think the balancing of these factors is of the greatest importance for success in the production of electrolytic zinc.

### CONCENTRATING SMELTING

As pointed out above in the discussion of ore supplies, ores of the Colorado territory are of a very complex character which do not admit of the high ratio of concentration or of the good recovery in the concentration itself. The cost of milling is relatively high and the recovery of all the metals contained is low as compared with the modern practice in simple ores.

The writer approached this problem, therefore, with the idea of eliminating to the greatest possible extent this preliminary concentration, as a great many of the ores of Colorado are already what one would consider, in other industries than zinc, relatively high-grade ores. Where the ores as mined contain too much waste, or are too low tenor in valuable metals, these can readily be concentrated to a product containing all the metals in a form which, though valueless to the retort process, are easily treated by the method proposed. That is, the ores may contain from 15 to 20 per cent. zinc with varying quantities of lead, silver and gold. It was necessary, therefore, to find a method for handling these ores:

1. Which would produce a high-grade simple concentrate, enabling us to carry such concentrate to a central refining point without incurring high freight charges.

2. Put this concentrate into such shape that it would be relatively easily separated into its constituent metals. Concentration smelting, with the volatilization of the zinc and the lead, was consequently decided upon.

Most of the ore supplies of central Colorado today are of sufficient grade to permit direct smelting in the blast furnace, whereby the principal part of the precious metal values become collected in a copper matte and the zinc-lead values are driven off as a fume and collected in the bag house. This substantially is the process as carried out at Florence, Colo., today.

It is evident that the physical character of the ores must be such as to permit charging into the blast furnace, or in the case of concentrate, that it must be put into this physical character. Again, that the ore charges must be balanced in such way as to give a slag that will carry the minimum of zinc to waste. Fortunately the Colorado ores are of such varied character as to permit of the fulfilling of both of these conditions quite readily.

The ores are charged without preliminary roasting; and with low copper contents on the charge a first matte is generally made carrying sufficient values to bear shipment directly to the refinery. Part of the silver in the charge is carried into the matte and the balance goes over with the volatilized metals into the bag house. The volatilization of the zinc and the cleanness of the slag still leave much to be desired and a great deal of interesting work in this direction remains to be done.

Concentrates produced in this way will contain only those metals and metalloids that are volatile under the conditions of the blast furnace, consequently it is a relatively simple mixture of oxide of zinc and sulphate of lead, with some sulphate of zinc and a small amount of silver.

In this connection there is necessary a word of acknowledgment of the pioneer work carried on on these lines by F. L. Bartlett in his smelter at Canon City, Colo., for a number of years.<sup>1</sup> A somewhat intimate acquaintance with Bartlett's work which the writer had the opportunity of obtaining some years ago really forms the ground work for our practice in Florence, Colo., today.

It is evident that in such concentrate as this the zinc is readily soluble in dilute sulphuric acid, and the lead and silver quite insoluble, consequently the purification problem becomes a very simple one. The physical character of the material is such as to make it easily handled for shipment and easily handled in the leaching vats. The metal contents are high, and shipment is in the direction of consumption of the metals, and this point must be borne in mind in connection with our refining at Keokuk.

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<sup>1</sup> *Trans.* (1893), 22, 661.

### *Residue*

The residue of lead and silver is relatively small when figured back on the tonnage of the original ore handled, and being a very finely divided, rather heavy material, it is with considerable ease washed reasonably free from zinc sulphate. The treatment of this for its lead presents no great difficulty, as the only metals of economic value are lead and silver.

### *Influence of Freight Rates*

It will be seen from the above that, considering the character of concentrate possible with the ores with which we have been dealing, and the location of the power that was available for the electrolytic process, only some such method as the above could be employed.

It is true that the loss of zinc in the blast-furnace slag is high, extremely high when compared with metallurgical work on other metals, but we endeavor as far as possible to make the milling and smelting operation one, so that one loss covers everything, and I am inclined to think that, under the conditions, to have a big hole in one pocket is perhaps no worse than to have little holes in three or four.

For work of this character it would, of course, be necessary that the smelting plant be located close to the ore supply. Such condition is easily attained in the Central Colorado territory, as the freight rates to the valley are relatively low, or a smelting plant could be located near the mines should it be necessary or desirable.

In the case of the ore supply of the San Juan territory, conditions are somewhat different, but it is rather fortunate that most of the ores as mined in the San Juan district are of a somewhat higher grade and would lend themselves readily to a rough concentration before shipment.

The freight on the zinc concentrates from the smelting point to the refinery becomes relatively small when figured back on the original ore from which it is produced and in any event the concentrate is moving toward the market for final metal production. Should such concentrate be moved in any other direction, the freight charge would become a direct charge on the cost of production which would not be redeemed in any way.

### *Cost of Current*

The cost of electric current in the production of electrolytic zinc is fortunately directly proportional to the metal recovered, rather than to the tonnage of ore treated. For this reason it becomes profitable to use it on a 20 per cent. ore or concentrate from Colorado, where it would not be profitable to use it under present conditions on a 60 per cent. concentrate from the Joplin district.

And, further, when the actual cost of current per ton of ore is figured back on a 20 per cent. ore it becomes, relative to the other charges incurred, only a small factor. In such case it would probably not exceed 25 per cent. of the whole cost of producing the metal and shipping to point of consumption, when figured on the present power cost in most of the eastern territory.

The actual power consumed in the production of metallic zinc will be for most cases about  $1\frac{1}{2}$  kw.-hr. per pound of zinc in the electrolytic tanks. Additions to this, of course, must be figured in the auxiliary power for the plant, transformer losses, rectifier losses, conductor losses, etc., all of which will be perfectly familiar to any electrochemist, so that altogether, probably, when working on central-station current 75 per cent. of the current purchased will go into the electrolytic zinc on the basis above mentioned.

Costs on this basis need not be figured, as each district will be a law unto itself, and power contracts will be based on load factor as long as central-station power is used. Central-station power is usually received as high-tension alternating current, and for the electrolytic tanks has to be stepped down and rectified. For the auxiliary power it has to be stepped down and may or may not be rectified, as the purchaser pleases.

Load factors in electrolytic work are usually very high, approaching 100 per cent., consequently the horsepower per year cost can be figured almost directly into a kilowatt-hour cost; that is 0.1 c. per kilowatt-hour is approximately equal to \$6.50 per horsepower year at 100 per cent. load factor.

### *General Conclusions on Concentrating Smelting Processes*

From the above rather crude analysis of the various factors entering into the production of electrolytic zinc, it will be seen that practically every district becomes a subject of study of itself. There are certain fields more than usually favorable for concentration, roasting and direct solution of the zinc.

This involves an ore easily concentrated, a concentrate relatively simple, having its zinc value high and preferably low in copper and low in iron. The more nearly these conditions are attained the more probable will be ultimate success in the work.

This requires, also, either close proximity of power to the ore supply or such a high-grade concentrate as to bear carriage to the power supply. We have seen that the ores of the Northwest particularly lend themselves to these conditions.

On the other hand, low-grade ores, ores difficult to concentrate, giving a complex concentrate high in iron or copper, or both, and power at a long distance from the orebodies, would suggest concentrating smelting.

Such conditions we have in the Colorado territory when considered in connection with power developments in the Mississippi valley.

The remarks concerning ore supply include those concerning power. Either the power must be convenient to the ore supply or as the ore moves toward the power it must also move toward the metal market. Again, we have the case of Northwestern power favorably located with reference to its ore bodies, and Mississippi Valley power favorably located with reference to metal market. In neither case will transportation charges accrue to a too great or unredeemed extent.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Zinc Dust as a Precipitant in the Cyanide Process

W. J. SHARWOOD, LEAD, SOUTH DAKOTA

(St. Louis Meeting, October, 1917)

IN the cyanide process, gold and silver are dissolved from crushed ore as double alkali-metal cyanides, from which they may be precipitated by such positive metals as sodium (amalgam), aluminum, or zinc, or by electrolysis. Two extreme conditions may be noted. Some works, especially slime plants practising decantation, use a relatively large volume of solution—possibly 4 or 5 tons per ton of ore—nearly all of which may require precipitation, so that the solutions handled are of much lower value per ton than the ore. On the other hand, in some leaching plants it is possible to extract with very little solution, and to percolate some of this more than once through the charge before precipitation, so that the solution to be precipitated may be much less than half the weight of ore, and proportionally richer. Solution intended for further use need not have all its precious metal removed, but any that has to be thrown away should be impoverished as far as is economically possible.

In spite of certain advantages possessed by other precipitants, zinc in some form has been almost universally used. In some of the first attempts to utilize cyanides as gold solvents, a "piece or plate of zinc" was suggested as a precipitant, but extension of surface was early recognized as a desideratum.

Macarthur and the Forrests adopted a "metallurgical filter" of zinc shaving, turned from disks or rolled sheets. They had previously experimented on other forms of zinc, and Macarthur records having tried zinc dust, which had of course been long known as a general reducing agent. It had also been known and used as a precipitant of precious metal from plating and photographic solutions, and comminuted zinc had been patented for recovering copper, etc., from ore leaches. Other inventors proposed the virtual making of zinc powder by the attrition of balls of zinc, and by similar means, during the passage of a stream of gold-bearing solution.

Sulman claimed the use of zinc dust, or fume, in a special apparatus, effecting a more or less regular feed of dust and solution by means of intermittent siphons; the mixture or "emulsion" (as it is still called) rising with diminishing velocity through an inverted cone in which it

deposited most of its burden, and being clarified by passing a baffle-box and finally a cloth filter. This system was used at Deloro, Ont.

The first cyanide plants in the United States to use zinc dust were the Mercur (Utah), treating coarsely crushed oxidized or roasted ore, the Drumlummon (Marysville, Mont.) and Delamar (Idaho) tailing plants, followed by the Homestake Sand Plant No. 1 (South Dakota). All these treated large tonnages and used filter-presses to catch the precipitate; the first three, and the Homestake when first installed, used square presses of the Johnson type, but differing in size and design. Nearly all the larger cyanide installations on the American continent now precipitate with zinc dust, and the Merrill triangular press has almost entirely superseded other forms for this purpose. Zinc shavings are still used almost exclusively in South Africa, and in the smaller plants elsewhere. Aluminum dust is used to a limited extent, chiefly on Canadian silver ores.

#### *Zinc-dust Precipitation*

The early method of applying zinc dust was to fill a vat with pregnant solution, agitate, add the dust and pump through the filter press. The zinc was first introduced by stirring the body of solution mechanically and sprinkling the dust on the surface. A variation was the use of compressed air at the vat bottom, through a small central cross or coil of perforated pipe. In a few seconds this set the solution in violent motion, and the zinc dust was then scattered on it with a shovel. This was at best a dusty and disagreeable job, and the introduction of oxygen at the precipitating stage was opposed to chemical theory. Elimination of the air, however, and cautious sifting of the zinc dust over the surface before pumping, showed imperfect precipitation—for instance, some tests gave only 75 per cent. of the gold precipitated as against 96 per cent. with thorough agitation by air.

To obviate the use of dry dust, various attempts were made to add it in the form of an "emulsion" or suspension in water or in cyanide solution.

An inverted 3-ft. iron cone was fitted with a short valved hose at the apex; the base was covered with a plate having a covered handhole, two valved inlets for compressed air, one for solution, and a relief valve. This pressure cone was two-thirds filled with solution, followed by zinc dust (say 40 lb. for a 200-ton tank); compressed air was then admitted by an inlet leading nearly to the apex to stir the mixture while adjusting the cover. The relief valve was then closed and the "emulsion" sprayed through the hose on the surface of the pregnant solution, which had meanwhile been stirred with air in the usual way.

At one plant such precipitate as settled on the tank bottom was allowed to accumulate there, while only the suspended portion went to the filter-press. Two grades of precipitate were thus produced, the settled material being decidedly lower in value and hard in texture.



Elsewhere, when the solution was pumped nearly to the tank bottom, a man entered the vat in rubber boots and swept settled material toward the pump intake, a small amount of fresh solution being used to assist the sweeping out. In spite of this sweeping there was a tendency for hard lime-zinc scale to accumulate on the bottom, and to some extent on the tank staves, so that after 6 months there might be several thousand dollars thus tied up in a pair of large vats, only recoverable by periodical "scaling" in which hammers and chisels were used.

Pumping a large tank to the press might occupy 2 to 6 hr., and one might expect re-solution to take place during this period when the zinc was added all at once at the start. Repeated tests showed that, although the precipitation by mere agitation was far from perfect, very little re-solution took place during pumping, but the part played by the zinc accumulated in the press was evidently an important one.

The amounts directly precipitated ranged from 20 to 90 per cent., while after passing the press 92 to 98 per cent. of both precious metals had been removed, but only 1 to 4 per cent. of the copper. Recently the positive removal of oxygen from the entire body of pregnant solution before adding zinc has been carried out on a working scale. Long-continued tests show that, by subjecting the solution to a vacuum during pumping, a considerable economy is effected, both in the amount of zinc dust consumed, and in the acid required for the subsequent refining. Patent has been applied for in connection with this modification of the process.

Turbid and very cold solutions, extremely low in alkalinity and free cyanide, present the most unfavorable conditions for precipitation. The quantity of zinc dust required per ton is not proportional to the precious-metal content, depending largely on the amount used in side-reactions.

### *Continuous Precipitation*

In adding zinc dust to a "moving stream," as it is termed in the Merrill patent, a uniform feed is absolutely essential to secure maximum efficiency, the volume added per ton being often so small that any diminution becomes temporarily fatal to precipitation. Feeders of various forms have been designed and used. One of the first and most satisfactory consists of a slow-moving horizontal belt, on the level surface of which is spread a charge of zinc dust in a layer of uniform width and thickness. The zinc usually falls into a small mixing cone, through which an auxiliary stream of solution passes, carrying the zinc by a small pipe to the pump intake. A slow drip of lead acetate or nitrate, or of strong cyanide solution, may be added here to facilitate precipitation. Many ingenious elaborations have been devised to secure uniformity in the fall of zinc and flow of auxiliary solution and chemicals. Formerly

the zinc and auxiliary solution in the mixing cone were continuously agitated by a small jet of compressed air. According to data published by Clark<sup>1</sup> a saving of about one-third of the zinc was effected by eliminating this air agitation of the zinc feed, and preventing air from being drawn into the mixing cone.

Another good feeder has a cylindrical roller or pulley slowly revolving at the lower end of a hopper, conveying a narrow ribbon of dust, the thickness of which is controlled by an adjustable slot. Or an auger-like horizontal screw may remove the zinc from a similar hopper. All such hopper-fed devices require jarring mechanism to prevent the zinc dust from bridging. A miniature tube mill has been introduced as a mixer to smooth out irregularities in the zinc feed.

Bosqui's zinc-dust feed system has two relatively small tanks, alternately filled and emptied by means of a tilting launder which actuates a counter and throws a measured charge of zinc to the tank to be filled; the zinc is continuously stirred by a set of jets of solution on a revolving agitator.

Mills has a set of vacuum filter frames submerged in a vat through which the mixture of zinc and solution is circulated by a centrifugal pump, while settling is overcome by revolving rakes.

### *Filter-Pressing*

Solution intended to be precipitated and thrown away may be run by gravity to a press at a low point; if for further use, it is generally pumped to a press at a considerable elevation, from which it falls to a storage tank. If the distance between intake and press is considered insufficient to allow of complete reaction, it is increased by leading the pipe in a zigzag line. A zinc press with gravity feed, the clear effluent elevated by a pump to storage, has the advantage over a pump-fed press in that zinc and precipitate are kept out of the cylinders and valve chambers; the claim is, however, made that the pump feed gives better precipitation with a given proportion of zinc.

A close filtering medium is necessary to retain zinc dust and the extremely fine precipitate obtained from low-grade gold solutions. In early practice the medium was sometimes paper between two cloths, or a single thickness of chain-cloth—a rather expensive fabric. Chain-cloth or cheaper heavy canvas may be covered with a light cheap twill, which is removed and burned at each cleanup to recover adhering precipitate, while the heavy backing is occasionally washed or treated with hydrochloric acid to remove lime accumulations. Another plan is to use two thicknesses of medium-weight cotton twill, the outer being taken off at

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<sup>1</sup> *Mining Magazine* (April, 1910), 4, 289.

each cleanup and either washed or burned. The other then becomes the outside cloth and a new or washed cloth is put under it for the next run.

One square foot of net filtering surface for 1.5 tons (say 50 cu. ft.) solution per day, or 6 tons per hour for 100 sq. ft., may be taken as a conservative ratio: with clear solutions a press may be run at double this rate for long periods. Colloidal suspended matter soon increases the pressure and reduces the pumping rate.

Two-inch distance-frames are suitable for a press used on gold solutions, 3-in. or 4-in. for silver. When a press is opened, most of the cake readily falls into the wheeled tray placed beneath and the remainder is removed by scrapers. It is normally soft but sometimes caked hard as a result of oxidation or the presence of calcium carbonate; this condition can often be controlled by excluding air from the solution.

It was early found advisable, and is still the custom, when starting up a zinc press after a cleanup, to add to the first charge 50 per cent. or more zinc in excess of the weight normally required; this is gradually diminished in successive charges, until the regular amount is reached. If the zinc feed is out too low at any time, no effect may be noted until several charges have been thus treated, then the "barren" assays rise suddenly and it becomes necessary to add a considerable excess for several charges, until normal working is restored. Apparently a certain excess of zinc must be maintained in the press, or re-solution takes place there to some extent. This is evident when a press stands idle for several hours; the effluent samples caught during the first few minutes of pumping will be abnormally rich—occasionally richer than the pregnant solution entering the press.

### *Fineness of Zinc Dust*

The virtue of zinc dust as a precipitant is explicable by its fine state of division or, what amounts to the same thing, its extended surface. Waldstein's patent claimed that the zinc oxide, invariably present, formed a beneficial galvanic couple with the metal, while Sulman's process involved the preliminary removal of oxide by a solvent. In early practice it was noted that dust containing 1 or 2 per cent. of lead was more effective than purer samples, and this was confirmed by laboratory tests on synthetic alloys. Cadmium seems to have but little effect. That fine division is the main factor is indicated by the fact that the finest unoxidized metallic zinc, made by grinding sifted filings, and levigating in absolute alcohol, can be made even more effective as a precipitant.

To get some idea of working conditions in using zinc dust on the large scale, the particles may be assumed to be equal spheres of a diameter which we may take as 0.0001 in. One pound of this assumed zinc dust

will then contain 7544 million particles and will expose 1650 sq. ft. of surface, which is much greater than the surface of an equal weight of shaving. In precipitating gold, the practical minimum of zinc dust is probably between 0.1 and 0.2 lb. per fluid ton of 32 cu. ft., or 50 to 100 parts per million of solution. One-tenth of a pound, uniformly distributed through a ton of solution, would give some 13,650 particles per cubic inch, spaced at an average distance of about one twenty-fourth inch. Doubling the weight of dust per ton reduces the average distance between particles by about 20 per cent. If we could substitute particles of half the diameter, the same weights of zinc dust per ton would give eight times as many particles per cubic inch, at half the distance apart. These considerations make evident the desirability of obtaining zinc dust in a very fine state of division, and the bad effect of stray coarse shot or agglomerated masses. In strong solutions, rich in silver, it is probable that the economic limit of fineness would soon be reached; with gold solutions, it seems unlikely.

### *Efficiency of Precipitation*

Zinc dust is sometimes valued by the percentage of zinc actually in the metallic state, estimated by the reducing effect on ferric sulphate or chromic acid; or the difference between total zinc and zinc as oxide may be taken as metallic. As a guide to its precipitating value this is insufficient, and the fineness, or the speed of reaction, must be considered. As a rule, a good zinc dust will nearly all pass a sieve of 200 meshes to the inch, and in some zinc dusts now obtainable very little over 1 per cent. is retained by a 300-mesh sieve. In either case, the coarser portion is merely accidental material—crystalline aggregates, a few shots, or foreign matter.

A laboratory test based on precipitating effect is carried out<sup>3</sup> by shaking half a gram of the dust with a solution containing an excess of double silver cyanide and a little free alkaline cyanide, and determining the silver precipitated. This is a valuable but not infallible guide in practice. Herz<sup>4</sup> has shown that, to obtain comparable results, extreme care is necessary in adjusting the free alkali and cyanide in the test solution.

A precipitating efficiency of 100 per cent. assumes pure zinc, 1 atom of which should precipitate 2 atoms of silver or its equivalent in gold or copper from the double cyanide. At this rate 1 unit weight of zinc

<sup>3</sup>W. J. Sharwood: Zinc-dust Tests. *Journal of the Chemical, Metallurgical and Mining Society of South Africa* (February, 1912), 12, 332; *Mining and Scientific Press* (May 11, 1912), 104, 659.

<sup>4</sup>*Trans.* (1915), 52, 139.

should precipitate 6.03 units of gold, 3.30 of silver, or 1.93 of copper, but in actual practice the results are much lower. Applied to commercial dust, the laboratory test usually shows an efficiency value of 30 to 60 per cent. Some extremely fine and pure "artificial" zinc dusts, prepared by re-distillation of spelter, have shown laboratory efficiencies of 75 per cent. or more, and pure electrolytic zinc powder is claimed to be equally good. Otherwise the efficiency is generally higher when lead is present to the extent of a per cent. or two. Distinctly coarse or granular zinc preparations generally show very low results.

Judged by the actual precious metal precipitated, the working efficiency on a large scale may be a mere fraction of 1 per cent. in the case of low-grade gold solutions. A considerable amount of zinc is always wasted in side-reactions, such as the evolution of hydrogen, reducing dissolved oxygen, or precipitating copper and lead. With rich silver solutions the efficiency may approach 50 per cent., especially if no attempt is made to recover the last traces of silver.

A practical example of mixed precipitation may be taken from the records of the Drumlummon tailing plant, covering three seasons, or 24 months of treatment of sandy tailing by leaching, the silver being largely in excess of the gold.

|                               |                                     |
|-------------------------------|-------------------------------------|
| Sand treated                  | 290,000 tons                        |
| Solution precipitated         | 390,000 tons                        |
|                               | Lb. Av.                             |
| Precipitate obtained          | 104,000                             |
| Zinc dust used                | 113,600                             |
| Metallic zinc in dust         | 102,240                             |
| Zinc remaining in precipitate | 40,300 = 39.4 per cent.             |
| Zinc dissolved                | 61,940 = 60.6 per cent.             |
| Gold in precipitate           | 2,380 = 0.39 per cent. efficiency.  |
| Silver in precipitate         | 16,000 = 4.75 per cent. efficiency. |
| Copper in precipitate         | 7,800 = 3.93 per cent. efficiency.  |
| Total efficiency (Au, Ag, Cu) | 9.07 per cent. efficiency.          |

Percentage efficiency is calculated by dividing the weight in pounds by the electrochemical equivalent (Zn 32.7, Au 197.2, Ag 107.88, Cu 63.5, and Pb 103.6, under these conditions) dividing the quotient by the number of equivalents of zinc used, and multiplying by 100. The metallic zinc in dust has been taken throughout at the approximate figure of 90 per cent.

Another instance may be taken from the published results<sup>2</sup> of a month's run (May, 1911) of Sand Plant No. 1 of the Homestake Mining Co., in which the silver is about 1 per cent. of the gold value. Some lead was added to the solution as nitrate and recovered in the precipitate.

The efficiency for gold was 1.18 per cent., for silver, 0.73 per cent., and for copper 0.19 per cent., in the "weak solution," making a total

<sup>2</sup> *Transactions, Institution of Mining and Metallurgy* (1912), **22**, 142, 149. Tables XV, XVI, XXI.

efficiency of 2.1 per cent. Similarly the total efficiency (Au, Ag, Cu) in the "low solution" was only 0.71 per cent. At the same time about 1 per cent. of the zinc was consumed in precipitating lead.

That such low efficiencies are tolerated in gold extraction is explained by the fact that, when "low solution" is going to be thrown away, it is obviously worth while to extract the last 2-c. worth of gold recoverable, if this can be done at the expense of 1 c. for zinc, making a fair allowance for refining cost. With zinc at 14.5 c. per pound, 1 oz. Troy costs 1 c. If this ounce is used to precipitate 0.001 oz. of gold (2 c.) from a ton of waste solution the practice is defensible, although the actual chemical efficiency attained is less than 0.002 per cent.

### *Zinc Dust vs. Zinc Shavings*

The practical efficiencies obtained with zinc dust have been, generally speaking, about the same as with zinc shavings, and the accumulation of zinc in the solutions is about the same; greater variations occur between two plants using the same process on different ores than between two plants using the different precipitants on similar ores.

The dust process involves a more expensive installation than zinc shavings, but has the advantage of greater compactness and cleanliness, and involves less labor in maintenance and cleaning up as well as less risk of theft. The periodical cleanup is absolute, while a holdover of several thousand dollars' worth of precious metal commonly occurs with zinc shavings, and makes it impossible to compare the actual with what is often called the "theoretical" recovery. After a destructive fire, precipitate in a filter-press has been found intact, while zinc boxes have entailed great difficulty in the attempts to recover their contents. At a gold plant a press occupying a floor space of 5 ft. by 14 ft. can easily carry a month's accumulation of \$40,000.

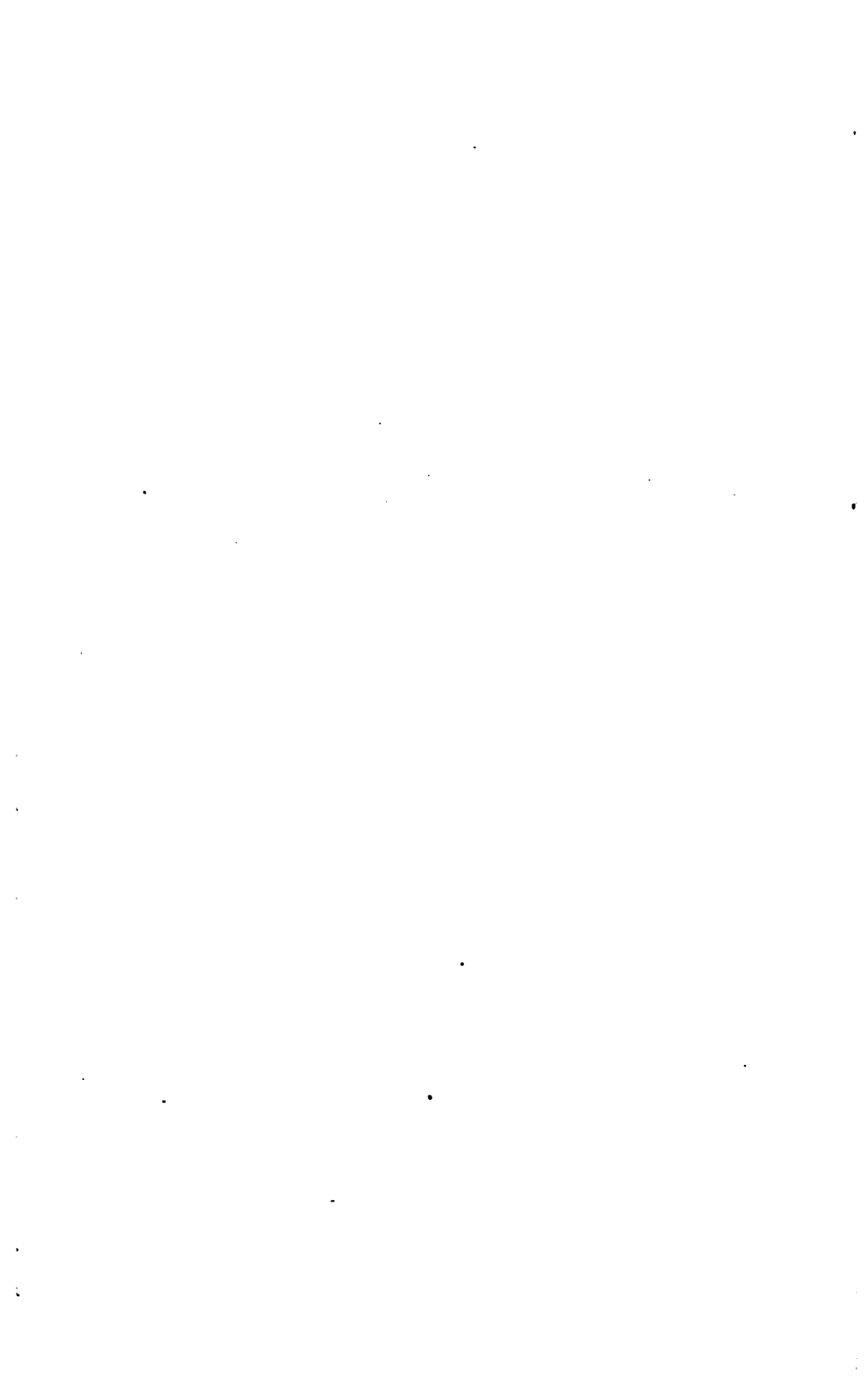
The comparative cost of the two systems at any time depends, of course, upon the wage scale and the relative prices of zinc dust and spelter.

### *Composition and Treatment of Precipitate*

Except in the absence of coarse fibers in the former, there is no essential difference in the composition of zinc-dust precipitate and that obtained with shavings. Both contain metallic gold and silver, usually co-precipitated as an alloy, sometimes amalgamated with mercury or alloyed with copper or with zinc itself—some zinc being apparently reprecipitated electrolytically with the precious metal. Any lead in the solution is also thrown down, while any lead and cadmium in the zinc dust remain with the undissolved zinc, which may form 20 to 70 per cent. of the dry weight of precipitate. Calcium carbonate, moisture, and fine

ore particles make up the total, with sometimes calcium sulphate or zinc ferrocyanide and basic cyanide. Lead, mercury, and silver may be present either as metals or sulphides, the latter condition resulting from sulphur compounds in the solution. Blowing air through the press to dry the precipitate causes rapid oxidation of zinc, and consequent heating. Dilute sulphuric acid, followed by thorough washing, removes most of the zinc and lime, and more or less copper, while lead, cadmium, mercury and silica remain with the precious metals. The mercury may then be removed by drying and heating in a retort with a little lime. Imperfect washing leaves zinc sulphate, and this, like calcium sulphate or sulphides, yields a matte in the subsequent refining. Hitherto no way has been found to utilize the zinc sulphate solution, impure and generally saturated with calcium sulphate, which is obtained in the acid treatment of precipitate.

Acid treatment, or lead refining and cupellation, or a combination of the two, is necessary before melting low-grade precipitate. Richer gold precipitate, and that from most silver ores, may be fused with a suitable flux, directly or after roasting, yielding fairly fine bullion.





DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## **Manganiferous Iron Ores of the Cuyuna District, Minnesota \***

BY E. C. HARDER,† PH. D., WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

### **INTRODUCTION**

IN view of the gradually decreasing known reserves of high-grade manganese ore and the rapidly increasing consumption of iron-manganese alloys in the steel industry, it is well to turn our attention to the development of low-grade manganiferous ore deposits and to the utilization of these ores in steel manufacture. The dependence of this country on imported manganese ore and ferromanganese has been forcibly brought to the attention of steel makers during the last few years by the cessation of manganese-ore imports from Russia, the restriction of manganese-ore imports from India and of ferromanganese imports from Europe. Should the importation of manganese ore from Brazil cease, the steel industry of this country would face a serious situation. The present practice of steel making requires ferromanganese in large quantities and ferromanganese of the grade used at present can be produced only from high-grade manganese ores such as the United States appears to possess only in insignificant amounts. There are in the United States, however, large reserves of manganiferous iron ore which could probably be used extensively in the steel industry if certain changes in the practice of steel manufacture were made. Thus high-grade manganese ore could be conserved. Among the most important deposits of manganiferous iron ore in this country are those of the Cuyuna iron-ore district of Minnesota. It is probable that in case of restriction of foreign supplies, the users of manganese in the United States would have to rely to a large extent on the Cuyuna district to make up the shortage.

Manganiferous iron ore was found in association with iron ore in the

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† Geologist, U. S. Geological Survey.

Cuyuna district in many of the early drilling operations and the possibility of its utilization was in the minds of various explorers in the district for a long time before mining operations were begun. Many drill holes in the northern part of the district passed through iron-bearing formation containing varying amounts of manganese. The distribution of the manganese was found to be very irregular, however. In certain drill holes high-grade manganiferous iron ore, containing 25 per cent. manganese and over, was encountered, while nearby holes showed little or no manganese. Even in the same hole great variations were found in the manganese content of the ore. Ore with more than 20 per cent. of manganese was found to be irregularly intermixed with material containing 2 or 3 per cent. of manganese or even less. Generally the iron content was found to be high when the manganese content was low and *vice versa*, so that the combined iron and manganese content was fairly regular. In many drill holes, however, material was encountered which was abnormally high in silica and low in combined iron and manganese.

In view of this situation and of the lack of demand for manganiferous iron ore in the industries, the mining of these ores was discouraging for a long time, and iron ores containing an appreciable amount of manganese were considered almost worthless. In 1911, however, the shaft of the Cuyuna-Mille Lacs Iron Co. was sunk and in 1913 the first manganiferous iron ore produced in the district was shipped from this mine. In 1913 a second shaft, that of the Iron Mountain Mining Co., was sunk and a few carloads were shipped from this mine for experimental purposes during the same year. The production gradually increased in succeeding years and as the supply of manganese in the country diminished due to war conditions, and its price rose, the demand for Cuyuna Range manganiferous iron ores increased. During the coming season, such ores will probably be produced from 14 or 15 mines in the district.

While this increase of demand for Cuyuna Range manganiferous iron ore has been produced largely under abnormal conditions, it seems probable that its extensive use in the industries will result in certain changes of metallurgical practice and it is a live question whether on account of such changes the demand for these ores will continue when normal conditions are reestablished.

#### SITUATION OF THE CUYUNA DISTRICT

The Cuyuna iron-ore district is situated in the central part of Minnesota, in Aitkin, Crow Wing, Morrison, and Todd Counties. The main central portion of the district is 90 miles (145 km.) west of Duluth and 55 miles (88.5 km.) southwest of the western part of the Mesabi district. The productive area lies south of the Mississippi River in the central

and southern part of Crow Wing County, but extensions of the iron-bearing beds are known to the northeast as far as the central part of Aitkin County and to the southwest in Morrison County and in the eastern part of Todd County. In the part of Crow Wing County north of the Mississippi River, also, exploration work has shown the existence of iron-bearing rocks at several localities.

The productive part of the Cuyuna district is commonly divided into two sections known as the north and south ranges. The south range includes the portion of the district lying south of the Duluth-Brainerd branch of the Northern Pacific Railway which runs southwestward through the district. The north range lies north of the railway. The

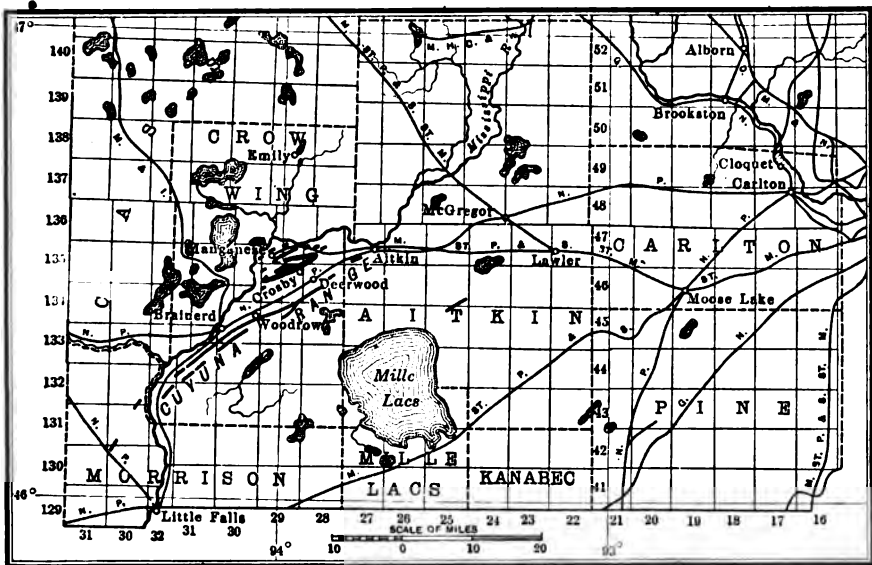


FIG. 1.—MAP SHOWING THE LOCATION OF THE CUYUNA IRON-ORE DISTRICT.

two ranges, while exhibiting certain differences in the character of the ore, are geologically and structurally similar. The south range with its extensions occupies a large stretch of territory. It has a known length northeast and southwest of about 65 miles (104.6 km.) and a maximum width of 3 or 4 miles (4 to 6 km.). Seven mines and exploration shafts have been sunk along it, of which four have produced ore, but of which only one, the Wilcox mine, has been in steady operation. Many undeveloped orebodies, the existence of which has been shown by drilling, are found along the south range. The north range occupies a comparatively small area, being about 10 miles long and 5 miles wide, but it contains most of the important mines in the district. It is in the northern part of the north range that the principal manganese iron-ore deposits

are found. There are at present 17 producing mines on the north range, while 7 or 8 new mines are being developed. Ten of the active mines have produced manganiferous iron ore. The north range is separated from the south range by a strip of territory from 2 to 3 miles wide in which iron-bearing rocks have not yet been found.

The iron-bearing rocks occurring in Crow Wing County north of Mississippi River are found mainly in the region of Emily and to the southeast between Emily and Aitkin. Ore has been shown to exist in this region, but no attempt has been made to mine it.

East central Minnesota is for the most part a low-lying region. The portion occupied by the Cuyuna district contains numerous low morainic hills interspersed with lakes and swamps. Extensive marshes occur in Aitkin County east and northeast of the district, while to the southwest and south are large outwash plains. The elevations in general range from 1150 to 1350 ft. (350 to 411 m.) above sea level, or from 548 to 748 ft. (167 to 227.9 m.) above the level of Lake Superior. Much of the area is under cultivation.

Two railroads serve the district, the Duluth-Brainerd branch of the Northern Pacific Railway and the Cuyuna Range branch of the Minneapolis, St. Paul and Sault Ste. Marie Railway. Up to the present time the latter has handled most of the iron-ore traffic.

## GEOLOGY<sup>1</sup>

The bedrock in the Cuyuna district is covered by a mantle of glacial drift which varies in thickness from 15 to 400 ft. (4.57 to 121.9 m.) Outcrops are found at only two places, (1) near Dam Lake and Long Lake

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<sup>1</sup> F. S. Adams: The Iron Formation of the Cuyuna Range. *Economic Geology* (1910), **5**, 729-740; and (1911), **6**, 60-70 and 156-180.

C. A. Cheney, Jr.: Structure of the Cuyuna Iron-Ore District of Minnesota. *Engineering and Mining Journal* (1915), **99**, 1113-1115.

E. C. Harder and A. W. Johnston: Notes on the Geology and Iron Ores of the Cuyuna District, Minnesota. *U. S. Geological Survey Bulletin* 660A (1917); Preliminary Report on the Geology of East Central Minnesota, Including the Cuyuna Iron-Ore District. *Minnesota Geological Survey Bulletin* (in press).

C. K. Leith: The Geology of the Cuyuna Iron Range. *Economic Geology* (1907), **2**, 145-152.

E. P. McCarty: Manganiferous Iron Ores of the Cuyuna Range. *Engineering and Mining Journal* (1915), **100**, 400-402.

C. R. Van Hise and C. K. Leith: The Geology of the Lake Superior Region. *U. S. Geological Survey Monograph* 52 (1911), 211 *et seq.*

Carl Zapffe: The Cuyuna Iron-Ore District of Minnesota. *Brainerd Tribune Supplement*, Sept. 2, 1910; Geology of the Cuyuna Iron-Ore District, Minnesota. *Mining World* (1911), **34**, 585-588.

Carl Zapffe and W. A. Barrows, Jr.: The Iron Ores of the South Range of the Cuyuna District, Minnesota (1912) **44**, 3-13.

in central Aitkin County south of the eastern end of the iron-bearing region and (2) at Randall in Morrison County near the southwestern end of the iron-bearing region. The outcrops at Dam Lake and Long Lake consist of quartzite and basic igneous rocks while those at Randall are composed of chloritic schist. Both have suffered metamorphism and deformation and are related to rocks which have been shown by drilling to occur in various parts of the Cuyuna district in association with the iron-bearing rocks.

The glacial drift in most parts of the district is sandy near the surface though locally there are areas of clay and loam. Swamps and marshes are numerous and in these localities the drift is overlain by beds of marl and peat. In the central part of the district several open-pit mines show good sections of the glacial drift. In these the prevailing succession from top to bottom is as follows:

1. Muck, peat and marl. Not everywhere present.
2. Yellow and gray sand. Forms main part of drift mantle. It varies in coarseness and locally contains thin pebble beds.
3. Gray stratified clay. Usually consists of fine, laminated clay and silt with boulders and pebbles which increase in abundance toward the lower part. It is called "hard pan" by the miners. Generally it rests on the bedrock surface but in a few places it is underlain by
4. Red clay with boulders.

Sand forms by far the most important part of the drift and locally comprises the entire thickness of glacial material. The gray clay is also widespread but shows great variations in thickness and distribution. The other phases are local. The average thickness of the glacial mantle over the main central part of the district varies from 50 to 80 ft. (15 to 24.4 m.)

The rocks of the Cuyuna district can be grouped under three classes; (1) metamorphosed sedimentary and igneous rocks interlayered with each other in beds and lenses, usually with steep dip due to close folding; (2) igneous rocks intruded into the metamorphosed rocks subsequent to their metamorphism and deformation; and (3) younger rocks which lie horizontally on the eroded surfaces of the preceding.

The age of the various rocks is not definitely known. The old metamorphosed rocks have been classed as upper Huronian on account of their general lithological similarity to and apparent areal continuation with the upper Huronian rocks of the Mesabi district.<sup>2</sup> They consist principally of schists and slates of various kinds associated with beds and lenses of iron-bearing formation. Locally quartzite has been found and at one or two places limestone has been reported to occur. Meta-

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<sup>2</sup> C. R. Van Hise and C. K. Leith: *The Geology of the Lake Superior Region. U. S. Geological Survey Monograph 52* (1911), 211 *et seq.*

morphosed igneous rocks, many of which have been rendered schistose but some of which retain the original textures, are common.

The schists are principally quartzose and argillaceous sericitic schists, but locally chloritic schist is very abundant. Gray, finely micaceous phyllite is also common. Of the slates two principal kinds have been met with in the drilling: (1) dark gray to black graphitic slate, and (2) laminated grayish-green or green slate. The slates and schists occupy the areas between the iron-bearing belts, of which seven or eight trend north-eastward through the district. They form the wall rocks of the ore-bearing beds or are interlayered with the ore-bearing rocks. The types most commonly found near the iron-bearing formation are green chloritic schist and laminated green slate. Graphitic slate, however, is also a common associate of the iron-bearing rocks. Where these rocks occur in close proximity to the iron-bearing formation they are generally heavily impregnated with iron and are dark red in color. They are then designated as ferruginous slate, ferruginous schist or paint rock. The other classes of old metamorphosed rocks are somewhat less abundant than the slates and schists. Quartzite and limestone probably occur only as small local lenses. Metamorphosed igneous rocks, however, and beds and lenses of iron-bearing formation are common and widespread. Metamorphosed igneous rocks which have in part been rendered schistose but in which igneous textures are still closely shown are especially abundant along the south range. They border beds of iron-bearing rock or occur interlayered with iron-bearing formation and slate. Most of them now consist of feldspar and chlorite. They were probably originally basic or semi-basic in character. Green chloritic schist, common both in the north range and south range, is believed to be an igneous rock so highly metamorphosed that the original textures have disappeared.

The iron-bearing formation of the Cuyuna district consists in its upper oxidized portion of ferruginous chert, ferruginous slate, iron ore and manganiferous iron ore, and in its deeper, unoxidized portion of cherty and slaty iron carbonate rocks. Amphibole-magnetite rock and magnetitic slate formed by the metamorphism of the iron carbonate rocks are found locally in both the upper and lower portions locally. In certain parts of the district both the original and oxidized phases of the iron-bearing formation are manganese-bearing. The former in many places contains as much as 3 or 4 per cent. manganese and the latter 30 per cent. or more.

The iron-bearing rocks are found as beds and lenses interlayered with the schists, slates and associated rocks. They occur in a number of parallel belts, but it seems probable that this duplication is due largely to intense folding and that only one main horizon of iron-bearing rocks exists. Both above and below this main horizon, however, there are locally smaller and less continuous lenses of iron-bearing rock separated

from the main formation by schist and slate. The main formation itself also in many places contains interbedded schist and slate which split it up into several beds, thus making the general occurrence very irregular.

The depth of oxidation of the iron-bearing rocks varies greatly in different parts of the district. In places the unaltered rock has been encountered less than 100 ft. (30.5 m.) below the bedrock surface while elsewhere oxidized phases have been found at a depth of 650 ft. (198 m.) or more below bedrock surface. Iron ore of good grade is known at this depth, indicating that locally the action of surface waters has been very extensive. In general, it appears that on the north range the oxidized iron-bearing rocks extend to greater depths than they do on the south range.

While most of the ferruginous chert, ferruginous slate, and ore in the Cuyuna district is doubtless derived from original ferrous carbonate rocks by weathering processes, it is not impossible that some of these rocks may have been originally deposited in their present form. Such rocks, of course, do not have the same limitations as to depth as the oxidized rocks have. The slates, schists, and other rocks of the district also show the effect of weathering for considerable distances below the bedrock surface locally, but not to the same degree as the iron-bearing rocks.

#### STRUCTURE OF THE ROCKS

The rocks of the Lake Superior region have suffered folding at several periods in their history. As far as is known, however, the forces causing the deformation at the different periods have been applied in approximately the same directions, *i.e.*, northwest-southeast, so that the results as apparent in the rock structure are superimposed upon each other. The final outcome of these various deformational activities has been that all the pre-Cambrian rocks of the Lake Superior region have been folded into a great complex synclinorium trending approximately northeast-southwest.

The rocks of the Cuyuna district are folded into a complex series of northeast-southwest trending anticlines and synclines.<sup>3</sup> The dip of the limbs of the folds is usually vertical or very steep and may be either to the southeast or northwest, the former dip predominating. Over considerable areas close folding has resulted in producing an approximate parallelism of both limbs of a fold. This is probably the explanation of

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<sup>3</sup> C. R. Van Hise and C. K. Leith: *The Geology of the Lake Superior Regions. U. S. Geological Survey Monograph 52* (1911), 620 *et seq.*

Carl Zapffe: *The Cuyuna Iron-Ore District of Minnesota. Brainerd Tribune Supplement*, Sept. 2, 1910.

C. A. Cheney, Jr.: *Structure of the Cuyuna Iron-Ore District of Minnesota. Engineering and Mining Journal* (1915), 99, 1113-1115.

the great predominance of southeast dips in the district, which are characteristic both in the north and south ranges. The pitch of the folds is usually very low and may be either to the northeast or to the southwest. Pitches in one or the other direction usually predominate in different parts of the district. Thus at the southwestern end of the north range the folds in general pitch to the northeast, while along the central part of the north range they pitch to the southwest. Along parts of the south range, the folds pitch so slightly that the crests of anticlines are practically horizontal for miles. Because of this low pitch of the folds the various rock layers appear on the erosion surface as approximately parallel northeast-southwest trending bands. Locally, where the pitch brings ore-bearing bands below the erosion surface, sharp turns occur and the layers double back on themselves. Elsewhere minor drag folds cause local irregularities in their trend.

On account of the lack of exposures and as yet insufficient drilling and underground development, it has not been possible to work out the details of the structure. Drag folds and other secondary structures point to the existence of several major and many minor folds in both the north range and the south range. Ultimately, by means of these structures, it may be possible to determine the relationship between the rocks in different parts of the district. The apparent absence of a definite stratigraphic succession over any considerable area, however, is a serious drawback in working out the geologic relations.

The general distribution and structure of the rocks in east central Minnesota indicates that the Cuyuna district is situated near the axis of the southwestern extension of the Lake Superior synclinorium. The close folding such as exists in this district would thus be naturally accounted for as well as the very gentle folding along the Mesabi range which is supposed to be on the north limb of the synclinorium. The south limb, which in Michigan and Wisconsin is formed by the rocks of the Penokee-Gogebic district, is as yet unknown in Minnesota. The extensive areas of granitic rock in the St. Cloud and Mille Lacs Lake regions may be south of the synclinorium or, as believed by Zapffe,<sup>4</sup> they may represent intrusions within it. It is known that some of these granites do intrude metamorphosed sediments, but the age of the latter is not definitely established. If the granite is lower-middle Huronian or Laurentian in age and represents the basement upon which the Cuyuna metamorphosed sediments rest, then the south limb of the synclinorium may be expected to run northeastward from Little Falls through the northern part of Mille Lacs Lake toward Kettle River. On the other hand, if the granite is of later age than the Cuyuna series the south limb of the synclinorium may be far to the southeast.

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<sup>4</sup> Carl Zapffe: *op. cit.*



## IRON- AND MANGANESE-BEARING ROCKS AND ORES

*Distribution*

As has been mentioned, seven or eight main belts of iron- and manganese-bearing formation run in an approximately northeast direction through the district. Those which have so far been found to contain orebodies of commercial importance are largely confined to the area south and east of the Mississippi River. However, some orebodies have been found in the iron-bearing areas in Morrison County west of the Mississippi River and in those in Crow Wing County north of the Mississippi River. As yet no attempts have been made to develop them. The northern of the iron- and manganese-bearing belts south of the river are included in the north range and the southern belts in the south range.

The most northerly of the north range iron- and manganese-bearing belts is only indefinitely known. Traces of iron-bearing formation occur in section 25, T.47 N., R.30 W., and thence northeastward in sections 20 and 22, T.47 N., R.29 W. beyond which it crosses the Mississippi River. No important orebodies have been found along it.

The next iron- and manganese-bearing belt to the south is of considerable importance, containing the manganiiferous iron-ore bodies of the Ferro and Algoma (formerly the Hoch) mines in sections 32 and 33, T.47 N., R.29 W. and other important orebodies in section 28 near Manganese and in sections 22, 27, and 23, T.47 N., R.29 W. in the vicinity of Island Lake. Certain breaks occur in the continuity of the belt, but in general it is fairly well known because of extensive exploration. The iron-bearing formation appears to be more or less manganiiferous throughout. It is bounded on the north mainly by green slate and on the south by graphitic slate.

South of this belt of manganiiferous iron-bearing formation and ore there are a number of scattered occurrences of iron- and manganese-bearing formation in the region north and west of Menomin Lake and southwest of Rabbit Lake, such as those in sections 34 and 35, T.47 N., R.29 W. The relation between the separate areas is not yet known. To the east of them, south of Rabbit Lake, is the important iron-bearing belt containing the Kennedy orebodies. This belt is known from the southern part of section 25, T.47 N., R.29 W. as far east as the eastern part of Rabbit Lake where it either dies out or turns to the south and doubles back on itself.

South of the eastern end of the Kennedy iron-bearing belt is the northeastern end of another iron-bearing belt which lies to the south. Whether these two belts are actually connected by a synclinal turn is not yet possible to say. The southern belt runs southwestward through the northwestern part of section 32 and the centers of section 31, T.47 N., R.28 W.

and section 1, T.46 N., R.29 W. into the broad belt of iron- and manganese-bearing formation lying north and northwest of Ironton and Crosby in sections 2, 11, and 10, T.46 N., R.29 W. which contains the Mahnomen and Mangan No. 2 orebodies along the northern margin, and the northern Thompson, Armour No. 1 and Pennington orebodies along the southern margin. West of the Mahnomen and Pennington mines, the belt divides into two parts separated by a strip of slate and schist. The northern part continues a short distance southwestward into section 9, while the southern part extends southwestward along the north side of Blackhoof Lake in the southern part of section 9, where it contains the Hillcrest and other orebodies, and thence through the northwestern corner of section 16 and the northern part of section 17 into section 18, T.46 N., R.29 W. where the Rowe mine is located. Here it makes a synclinal turn northward and doubles back on itself for a short distance. The extent of the backward turn is not yet known.

This iron- and manganese-bearing belt is one of the most important in the district. It is known continuously over a length of more than 8 miles and along it are some of the most important ore deposits in the district. Unlike the belts to the north, this belt contains many important iron-ore deposits, though manganiferous iron ores also are found in it at various places. On the north, it is separated along the central portion from the Cuyuna-Mille Lacs-Sultana manganiferous iron-bearing area by a thin band of slate or schist, while on the south another thin band of schist separates it from the important Croft-Armour No. 2 iron-bearing belt.

The Cuyuna-Mille Lacs-Sultana manganiferous iron-bearing area is in section 3, south and southeast of Menomin Lake. It is manganiferous throughout and contains important bodies of manganiferous iron ore. It is known to be fairly wide near the Sultana and Cuyuna-Mille Lacs mines, but its longitudinal extent northeast and southwest is only imperfectly known. Areas of iron-bearing formation, however, occur north and south of June Lake in the northern part of section 9 and these may have some connection with the Cuyuna-Mille Lacs-Sultana area. Other orebodies occurring in the Cuyuna-Mille Lacs-Sultana ore-bearing area are those of the Mangan No. 1 and the Hopkins mines, the latter containing both iron ore and manganiferous iron ore.

The Croft-Armour No. 2 belt has been well explored to the northeast as far as section 1, T.46 N., R.29 W., where the Croft mine occurs. From this point it runs southwestward through the Meacham, Thompson, Armour No. 2, and Ironton properties in section 11, T.46 N., R.29 W., and continues a short distance into the eastern part of section 10, where it pinches out. The belt is fairly narrow throughout its length. North of the Armour No. 2 and southern Thompson orebodies which occupy its central portion in section 11, is a band of green schist which separates it from the Pennington-Armour No. 1 iron-bearing area.

The Croft-Armour No. 2 belt consists very largely of iron-bearing formation and associated iron ore, though locally, as in the eastern part of the Armour No. 2 property and the northern part of the southern Thompson orebody, manganiferous iron-bearing formation and ore occur in it. The proportion of ore to iron-bearing rock in the belt is high, the ore being largely a somewhat siliceous, hydrated, brown to red hematite, but in places containing important masses of red, medium soft, high-grade hematite, some of which is of Bessemer quality. The ore-bearing belt is bounded on the south mainly by green chloritic schist and red hematitic schist.

South of the Croft-Armour No. 2 and the Pennington-Rowe iron-bearing belts are local occurrences of manganiferous and non-manganiferous iron-bearing formation and ore, such as those in sections 19, 17, 12, and 1, T.46 N., R.29 W. These are the southernmost areas of iron-bearing formation in the north range.

The south range iron-bearing belts, although narrow, are far more extensive longitudinally than the iron-bearing belts of the north range. Of the known occurrences of south range iron-bearing formation those farthest northeast are in the Rice River region northeast of Aitkin. From this point southwestward as far as Deerwood many areas of iron-bearing formation are known. They are approximately along the same line of strike, but large unexplored areas are found between them. From Deerwood southwestward the south range iron-bearing formation has been traced more or less continuously to a point a short distance southwest of Barrows, a distance of about 24 miles. Mines and exploratory shafts are found along it at intervals, but many orebodies of importance remain undeveloped.

Although throughout most of this extent, the south range iron-bearing formation occurs as a single band, there are locally small stretches where two or three parallel bands occur. There are also numerous places along the belt where the iron-bearing formation has not yet been found and barren stretches occur which apparently break the continuity of the iron-bearing belt. To what extent these breaks indicate absence of iron-bearing formation and to what extent they indicate incomplete exploration has not been determined. South and southeast of Brainerd and south of Barrows a second iron-bearing belt has been shown to exist, running parallel to and about  $1\frac{1}{2}$  miles south of the main south range belt. Orebodies have been found locally along it.

The Morrison County belts of iron-bearing formation mentioned above are the continuations of the south range belts west of Mississippi River. Minor occurrences of iron-bearing formation are known in the vicinity of Bay Lake and Clear Lake south of the south range.

The areas between the different belts of iron-bearing formation have up to the present been only very imperfectly explored. However, in

almost every place where drilling has been done, some variety of schist or slate has been encountered, such as chloritic, sericitic, quartzose, or graphitic schist or slate. Green and gray micaceous and chloritic schist, green slate and black graphitic slate appear to be by far the most abundant. Locally quartzite has been encountered but the amount is usually small, and apparently it forms local lenses. Elsewhere considerable areas of different kinds of basic or semi-basic igneous rocks have been found.

The orebodies are found at intervals along the iron- and manganese-bearing formation belts. In some belts they are very abundant, whereas in other belts they are almost absent. Even along single ore-bearing belts they are grouped more or less in certain parts with stretches of barren iron- or manganese-bearing rock between. Thus along the Croft-Armour No. 2 belt, ore is almost continuous throughout, while in Armour No. 1-Pennington-Rowe belt important orebodies are grouped in certain localities with stretches of ferruginous chert and ferruginous slate between. Other belts, as the main south range belt, have orebodies scattered along them at irregular intervals separated by barren areas consisting of ferruginous chert, ferruginous slate, amphibole-magnetite rock, or magnetitic slate.

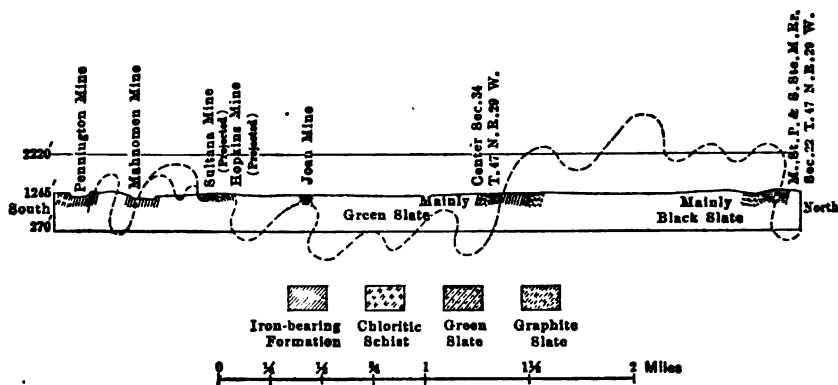


FIG. 2.—GEOLOGIC SECTION ACROSS THE NORTH RANGE, CUYUNA DISTRICT, SHOWING THE POSSIBLE STRUCTURE OF THE ROCKS.

The relation of the different iron-bearing and manganese-bearing belts to each other has not yet been determined. Several belts are known which represent the outcrops of opposite limbs of folds and it is believed that eventually all the different north range belts and possibly also the south range belts will be found to be thus related. Locally, it has been found that rocks between certain ore-bearing belts have characteristics which distinguish them from rocks between other ore-bearing belts. Thus in some areas graphitic slates predominate and elsewhere green slates or quartzose sericitic schists or phyllites predominate. Thus the relation between different ore-bearing belts may be determined by a

study of the associated rocks. Fig. 2 shows a north-south section through the north range with dotted lines to indicate the possible anticlinal and synclinal structures. The dotted lines should be regarded simply as a suggestion of what may be found.

### *Lithology and Occurrence*

The iron- and manganese-bearing formation of the Cuyuna district presents a variety of lithological types. Among the more common rocks composing the layers are hematitic and limonitic chert, hematitic and limonitic slate, cherty and slaty ferrous carbonate rock, green laminated slate, siliceous magnetitic slate, quartz-amphibole-magnetite rock, jaspilite, dark blue, red, brown, black and yellow iron ore, black, red, or brown manganiferous iron ore, green chloritic schist, and dark red hematitic schist.

Limonitic and hematitic chert are very abundant in the north range where they compose the principal part of the iron-bearing layers. They usually consist of white, pink, or gray chert interlaminated with hematite or limonite. The hematite or limonite laminæ may be pure iron oxide or they may be very siliceous. Frequently the iron oxide present in the laminæ is rather of the nature of a stain or an impregnation. Thus there are all gradations of ferruginous cherts from siliceous iron ore to chert containing only a small percentage of iron oxide. Frequently instead of being banded the ferruginous cherts present a blotched appearance due to the irregular distribution of chert and iron oxide or to the irregular staining of the chert by the latter. The more siliceous iron-stained chert usually shows a more irregular distribution of chert and iron oxide.

The chert laminæ in ferruginous chert are typically fine-grained, dense and flinty, but locally they have suffered partial disintegration and present a fine sugary appearance. When completely disintegrated they break up into fine sand, giving rise to what is known as wash ore. Wash ore is soft ferruginous chert from which the disintegrated chert can be removed by washing while the iron oxide remains behind and becomes concentrated.

The ferruginous chert occurring in the Cuyuna district is very similar to that which is found in most of the other Lake Superior iron-ore districts except the Mesabi. In all the Michigan and Wisconsin districts, the common rock composing the iron-bearing formation is a more or less regularly banded ferruginous chert which with the associated ferruginous slate is believed to have been derived by alteration from original cherty and slaty iron carbonate rocks. In the Mesabi district, however, the ferruginous chert is largely of the type known as "taconite," a rather massive, bedded cherty rock irregularly speckled with iron oxide which is supposed to have resulted from the oxidation of the ferrous silicate

rock "greenalite." No taconite or greenalite have been found in the Cuyuna district, although certain jaspilite layers have oolitic textures like taconite. It is, therefore, supposed that here, as in most of the other districts, the original rock from which the present hematitic and limonitic chert and slate and iron ore have in large part been formed is a banded cherty and slaty ferrous carbonate rock.

Hematitic and limonitic slate are not as abundant in the district as ferruginous chert, but occur locally as beds and lenses interlayered with ferruginous chert, iron ore and manganiferous iron ore. They are usually reddish, brownish, or yellowish in color and are finely laminated. Siliceous phases of the slates occur also which show gradation into ferruginous chert. These are thinly laminated but are hard and frequently contain thin chert layers. The ferruginous slates represent stages in the deposition of the iron-bearing formation when fine, argillaceous sediment rather than silica was being deposited with the iron. They are not as abundant in the Cuyuna district as in some of the other iron-ore districts, especially the Mesabi.

The ferruginous slates differ from the hematitic schist to be described later, in that they are of sedimentary origin and have been deposited with and suffered the same alterations as the ferruginous cherts. The hematitic schist, on the other hand, results from the alteration and impregnation with iron oxide of the green chloritic schist, a rock probably of igneous origin, which is commonly associated with the iron-bearing formation layers.

Cherty and slaty iron carbonate rocks are believed to be the original rocks from which the iron-bearing formation of the Cuyuna Range in its various phases is largely derived. Such rocks have been reported from many parts of the district in the deep drilling. Several different types occur with gradational phases between them. They vary in composition and texture and range from light to dark green or gray in color. Usually they consist of interlayered chert or argillaceous material with amphibole and iron carbonate. Magnetite is a common constituent and with increasing magnetite and recrystallization of the other constituents these rocks grade into amphibole-magnetite rock and magnetitic slate.

Cherty and slaty iron-carbonate rock generally shows distinct banding and many layers show fine lamination, the latter being characteristic especially of the slaty phases. The banding is largely due to interlayering of different constituents. The laminae of chert and siderite are usually light gray or greenish in color, while those of argillaceous material are darker and with increasing magnetite become almost black. Some phases of iron-carbonate rock are light gray in color and consist almost entirely of chert and siderite which may be intermixed, or they may be more or less segregated into banded layers. Grains of magnetite often occur along lamination planes, making the banding more marked.

Other phases of iron-carbonate rock contain considerable amphibole and these show interbanding of light gray and somewhat darker green layers. Still other phases are very dark gray or green and consist mainly of amphibole and argillaceous material mixed with more or less siderite but containing little or no chert. Such phases are usually very fine-grained and show no marked banding. All these various types of iron-carbonate rock occur interbedded with each other and grade into each other.

With increasing argillaceous material, slaty iron carbonate rock grades into green laminated slate. This is a light to dark green rock generally showing very perfect and regular lamination of light and dark layers and consisting principally of finely crystalline amphibole. Most of it has slaty cleavage which lies at varying angles with the lamination planes. Green laminated slate is interbedded with cherty and slaty iron-carbonate rocks. Upon oxidation and impregnation with iron it gives rise to dark red ferruginous slate.

The siliceous magnetitic slate and amphibole-magnetite rock are phases of the original iron-bearing formation which have suffered metamorphic alteration and recrystallization. They are banded or laminated rocks, usually very dark-colored with a tinge of green. They consist mainly of amphibole, magnetite, and chert or quartz. The fine-grained, finely laminated types with slaty structures are known as magnetitic slates, while the more coarse-grained and coarsely-layered types are known as amphibole-magnetite rock. The banding is due to the segregation of the different minerals into layers and also to a difference in the coarseness of crystallization. Thus layers of fairly pure magnetite alternate with layers of amphibole, chert, or quartz, or mixtures of these minerals. Some bands are very fine-grained, especially those consisting mainly of magnetite, while other bands, as those of amphibole, are often medium coarse-grained. In some phases the silica is medium coarsely crystalline (quartz), and in other phases it is microcrystalline (chert). The segregation of minerals in the amphibole-magnetite rocks and magnetitic slates is never perfect. Small amounts of fine disseminated magnetite usually occur scattered through the quartz, chert, and amphibole layers, while quartz, chert and amphibole occur intermixed with magnetite along the borders of magnetite bands.

The composition of these rocks shows a wide range. In places they consist largely of magnetite and amphibole and elsewhere they contain a considerable amount of silica with amphibole, but with little magnetite. Carbon in the form of graphite is locally present. Irregularities in the texture are shown by the thickening or pinching of laminæ. These irregularities, however, are much less marked than they are in the ferruginous cherts.

The amphibole-magnetite rocks and magnetitic slate of the Cuyuna district differ from the typical amphibole-magnetite rock of other Lake

Superior iron-ore districts, such as the Mesabi, Gogebic, and Marquette, in that they are fine-grained, more perfectly laminated, and that in general they contain less quartz and more ferromagnesian minerals. The typical amphibole-magnetite rock of the Cuyuna district is a finely banded rock consisting of alternating bands of magnetite and amphibole with more or less chert or quartz. That of the eastern part of the Mesabi district, on the other hand, is an irregularly banded rock consisting of alternating layers of fine-grained magnetite and coarse-grained quartz and amphibole. In the Cuyuna district, however, highly quartzose and cherty phases of the amphibole-magnetite rock and magnetitic slate are also common.

Jaspilite is of rather rare occurrence in the Cuyuna district, but well-defined beds of it are found in several of the mines. Thus at the Cuyuna-Mille Lacs mine both the hard, red-banded jasper and the specular, schistose jasper occur along the south wall of the orebody where they are interlayered between typical banded hematitic chert on one side and low-grade manganiferous iron ore on the other side. The Hopkins mine also contains hard dense jasper.

Jaspilite is metamorphosed and recrystallized ferruginous chert. Where the metamorphism has not been very intense, a hard, dense, bright red rock is formed. This is the more common phase, while the crystalline, specular form which results from pronounced recrystallization is more local in its occurrence. Some jaspilite layers show a well-defined oolitic texture. In places in the district, very hard, dense, siliceous ferruginous chert occurs to which the term jasper is sometimes applied. This rock, however, varies in color from brown or reddish-brown to black and is simply a very dense form of the ferruginous chert.

The green and gray chloritic schist and the dark red hematitic schist do not belong to the iron-bearing formation proper. They usually occur as small lenses within the iron-bearing formation layers or they may bound the iron-bearing layers on either side. The dark red hematitic schist usually occurs along the borders of such lenses where they are in contact with the iron-bearing rocks or iron ore and have become impregnated with hematite for a variable distance from the contact while the green or gray chloritic schist occurs in the center of such lenses or forms entire layers where they are not in contact with iron-bearing formation.

As has been mentioned, it is believed that the chloritic schists represent original masses of basic igneous rock which were intruded as sills into the iron-bearing formation and associated sedimentary rocks while these were still largely in an unmetamorphosed condition. Some of them may have originated as local surface flows. Subsequent metamorphism and deformation were suffered alike by igneous rocks and sediments.

The Cuyuna orebodies are as a rule roughly tabular in shape, the longer axes being parallel to the bedding of the inclosing rocks. As the



beds usually dip steeply, the orebodies are shown at the surface as bands which extend for considerable distances along the strike of the beds. They vary in width up to several hundred feet and are usually continuous, some of the known orebodies being more than a mile long. Their behavior with depth varies. Some are comparatively shallow, extending perhaps to a depth of 100 or 200 ft. (30 to 60 m.) below the base of the glacial drift, where they give place to ferruginous chert, amphibole-magnetite rock, or unaltered iron-bearing formation. Others extend to greater depths, either continuing directly downward along the bedding or running diagonally downward parallel to the strike in the form of shoots. The data regarding the depths reached by the orebodies is still very incomplete. The deepest mining operations at present extend about 250 ft. (45 m.) below the base of the glacial drift, and to this depth ore is known to continue uninterruptedly. The drilling in the district has for the most part been shallow, few holes occurring which reach a depth greater than 700 ft. Ore, however, has been encountered in some of the deeper holes at depths up to 650 ft. below the base of the glacial drift and it is supposed that certain orebodies go much deeper.

The orebodies are usually inclosed between walls of ferruginous chert, ferruginous slate, or other iron-bearing rocks. Where ferruginous chert forms the wall rock, the contact is usually very irregular. Many beds and irregular masses of ferruginous slate and chert occur in the orebodies. Green chloritic schist, green laminated slate, and dark red hematitic schist and slate are also common associates of the iron ore and manganiferous iron ore. They may form the wall rock of the orebodies or they may occur interlayered with the ore, more or less parallel to the bedding. Other rocks associated with the orebodies are amphibole-magnetite rock, magnetitic slate, ferrous carbonate rock, and black carbonaceous slate. These are especially common along the south range where they form the wall rocks in many places. In the north range also, they have been found in many places underneath and along the walls of orebodies representing the unaltered portions of the iron-bearing formation.

Most of the iron ore and manganiferous iron ore occurring in the orebodies of the Cuyuna district is soft, but hard ore is also abundant both in the north and south ranges, and is more or less irregularly associated with the soft ore. The Cuyuna ore probably shows a greater variety of texture, composition, and color than the ore from any of the other Lake Superior iron-ore districts. It is associated with many different phases of the iron-bearing formation and varies in character and appearance according to the kind of iron-bearing rock with which it is associated. The iron ore shows all stages of hydration from pure reddish-blue hematite to ochreous, yellow limonite and both argillaceous and siliceous phases are common. On the north range it varies from dark reddish-blue high-grade hematite to reddish and yellowish-brown siliceous or argillaceous

hematite and limonite, while siliceous and argillaceous manganiferous iron ore is of common occurrence. Some of the rich hematite ore is finely crystalline, but most of the ore, especially the more hydrated material, is amorphous. Locally, wash ore is associated with the other ores. On the south range the typical ores are reddish-brown hydrous hematite, dark brown to black limonite, and yellow, ochrous limonite.

The manganiferous iron ores consist of a mixture of various manganese oxides associated with more or less hydrated iron oxide. In most manganiferous iron-ore bodies local masses of pure black manganese oxide are found, some of it crystalline, but this ore is usually present in minor quantity and occurs irregularly intermixed with manganiferous iron ore. The manganiferous iron ores generally vary in color from dark red or brown to black and are usually fine-grained or amorphous.

### *Geologic Relations and Origin of Manganiferous Iron Ores*

*Occurrence.*—The manganiferous iron-ore deposits of the Cuyuna district are confined largely to the north range, although at two or three places along the south range iron ore carrying up to 3 or 4 per cent. of manganese has been found. Recently manganiferous iron ores are reported to have been found also in the region between Aitkin and Emily north of the Mississippi River. No manganiferous iron ore has been mined outside of the north range, however.

The principal deposits of manganiferous iron ore which have been found in the north range are those occurring along the northern iron-bearing belts south, east, and north of Menomin Lake and west of Rabbit Lake. Scattered deposits, however, have been found also along the southern iron-bearing belt extending from the Rowe mine northeastward to the Kennedy mine and locally in the area south of this belt. Six manganiferous iron-ore mines shipped ore in 1916 or previous years, these being, from north to south, the Algoma mine ( $NE\frac{1}{4}$  of  $NW\frac{1}{4}$  sec. 33, T.47 N., R.29 W.), the Ferro mine ( $SE\frac{1}{4}$  of  $NE\frac{1}{4}$  sec. 32, T.47 N., R.29 W.), the Cuyuna-Mille Lacs mine ( $SE\frac{1}{4}$  of  $SW\frac{1}{4}$  sec. 3, T.46 N., R.29 W.), the Mangan No. 1 mine ( $NE\frac{1}{4}$  of  $SW\frac{1}{4}$  sec. 3, T.46 N., R.29 W.), the Sultana mine ( $SW\frac{1}{4}$  of  $SE\frac{1}{4}$  sec. 3, T.46 N., R.29 W.), and the Mangan No. 2 mine ( $NE\frac{1}{4}$  of  $NE\frac{1}{4}$  sec. 10, T.46 N., R.29 W.). In addition, manganiferous iron ore has been shipped in appreciable amounts from the following iron-ore mines: Mahnomen mine ( $NW\frac{1}{4}$  of  $NE\frac{1}{4}$  sec. 10, T.46 N., R.29 W.), Thompson mine ( $W\frac{1}{2}$  of  $NE\frac{1}{4}$  sec. 11, T.46 N., R.29 W.), Armour No. 2 mine ( $S\frac{1}{2}$  of  $NW\frac{1}{4}$  sec. 11, T.46 N., R.29 W.) and Hillcrest mine ( $NE\frac{1}{4}$  of  $SE\frac{1}{4}$  sec. 9, T.46 N., R.29 W.). Thus out of a total of 21 mines which have produced ore in the Cuyuna district 10 have yielded manganiferous iron ore. At the present time, four or five new manganiferous iron-ore properties are being developed and

ore will probably be shipped from these in the near future. Among them are the Merritt mine (SW $\frac{1}{4}$  of NW $\frac{1}{4}$  sec. 33, T.47 N., R.29 W.), the MacKenzie mine (SE $\frac{1}{4}$  of SE $\frac{1}{4}$  sec. 28, T.47 N., R.29 W.), and the Joan mine (SW $\frac{1}{4}$  of NE $\frac{1}{4}$  sec. 3, T.46 N., R.29 W.).

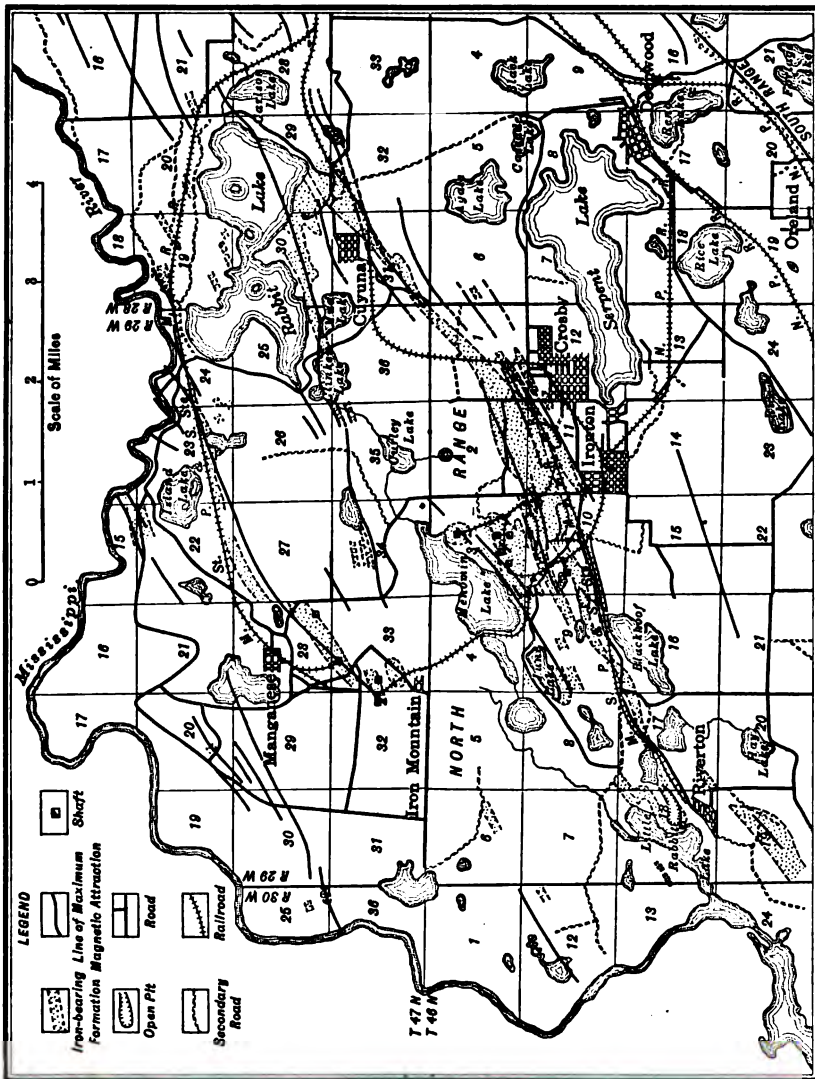


FIG. 3.—MAP SHOWING THE AREAS OF KNOWN IRON-BEARING FORMATION AND LINES OF MAXIMUM MAGNETIC ATTRACTION IN THE NORTH RANGE, CUTUNA DISTRICT, MINNESOTA.

In general occurrence, the manganiferous iron-ore deposits are similar to the iron-ore deposits, in fact there are many orebodies in the district in which both classes of ore are found. Like the iron-ore bodies, the deposits of manganiferous iron ore are roughly tabular in shape, the longer

diameters generally being parallel to the bedding of the formations inclosing them. They are, however, considerably more irregular in outline. Most of the manganiferous iron-ore bodies are associated with steeply dipping rocks and therefore dip at steep angles. Several deposits, however, have been found which occur along the axes of folds and these have a greater horizontal extent and are more irregular than bodies which occur on the steeply dipping limbs of folds.

Steeply dipping bodies of manganiferous iron ore are found in the Ferro, Algoma, and Mahnomen mines. In all these mines the general strike of the orebodies is approximately northeast and the dip averages between 60° SE and vertical. Although they have the same general attitude, there is a considerable difference in the form of the deposits in the three mines, due to the character of the material in which the ore occurs. At the Ferro mine the ore, which is of higher grade than most of the ore in the district, occurs along a comparatively narrow zone in manganiferous iron-bearing formation which is prevaillingly cherty. Within this zone the richer manganiferous ore is generally concentrated along the hanging-wall side. In the Algoma mine the ore-bearing rock is also manganiferous and ferruginous chert, but here the ore-bearing zone is much wider than that at the Ferro mine. Ore does not occur throughout the zone, however, but is confined to four or five layers or lenses separated from each other by lean ferruginous chert or manganiferous ferruginous chert. The ore in the different lenses varies in character. In one lens it is low in phosphorus but high in silica, in another it is high in manganese and also moderately high in phosphorus, while in a third it is high in iron and only moderately high in manganese. The Mahnomen orebody differs from those of the Ferro and Algoma in having ore scattered throughout a comparatively wide zone. The ore is found in red slate or in yellow to brown clayey material probably derived from red slate, and occurs in irregular nodular masses or as irregular replacements along bedding and fracture planes. It is found in greater or less amount throughout the ore-bearing zone, being richer in the clay bands than in the red slate bands. Thus in these three mines, while the attitude of the beds is similar, the nature of the orebodies is quite different.

In the Cuyuna-Mille Lacs and Sultana mines, the orebodies are found along the axes of folds. The north orebody at the Cuyuna-Mille Lacs mine occurs along the crest of a southwestward pitching anticline, while the south orebody at the Cuyuna-Mille Lacs and that at the Sultana mine are found in a southwestward pitching synclinal trough. In both of these mines the ore-bearing zones are considerably wider than those at the other mines mentioned, but the ore is somewhat more irregularly scattered through them and does not occur as distinctly along stratigraphic horizons. The ore of the Cuyuna-Mille Lacs north orebody, similarly to that at the Algoma and Ferro mines, is associated chiefly

with manganiferous ferruginous chert. The situation of the orebody along the axis of an anticline suggests that it may occupy a fracture zone, the ore having replaced the fractured rocks. However, in its downward extension the orebody divides into two limbs, one south of the axis of the fold and one north, thus suggesting that a certain stratigraphic horizon was more amenable to replacement than the rest of the formation. The Sultana orebody resembles that at the Mahnomen mine in that the ore is associated with red slate and yellow and brown clay. Although of considerable width, the greatest extent of the deposit is northeast-southwest parallel to the strike of the rocks. Along the south side the rock beds dip steeply northwest while along the north side they dip more gradually to the southeast. The ore is found irregularly distributed, occurring as replacements and veins in the slate and clay. The south orebody at the Cuyuna-Mille Lacs mine occurs southwestward, along the strike of the Sultana orebody, and is similar to the latter in structure. The associated rock, however, is in general more cherty than at the Sultana mine.

In other deposits of manganiferous iron ore in the district there is apparently no definite relation between the structure of the rocks and the form of the orebody. Thus in the Armour No. 2 mine the rock beds have a general steep dip to the southeast. The Armour No. 2 orebody, however, consists in its southwestern part of rich red and blue hematite, in the northeastern part of dark brown to black manganiferous iron ore. The line separating these two types of ore is fairly sharp and cuts diagonally across the strike of the rocks. In other orebodies, such as the Hillcrest, manganiferous iron ore is found as irregular masses surrounded by iron ore. In such cases it probably represents local replacements in iron-ore deposits.

The borders of the manganiferous iron-ore bodies are usually indefinite and the outlines are irregular. There are all gradations between ore and the manganiferous or non-manganiferous iron-bearing wall rock. In places this gradation extends over only a few inches or a few feet, but elsewhere irregular masses of lean rock are found scattered through the ore many feet from the outer edge of the orebody.

*Size.*—The Cuyuna manganiferous iron-ore bodies are smaller than the iron-ore bodies and usually do not extend to as great depths. Deposits are known which show ore practically continuously over a width of 300 ft. (91 m.) and which extend along the strike for 1500 ft. (457 m.) or more. Usually, however, they are much smaller. The greatest depth to which any of the deposits have been worked is 175 ft. (53 m.) below the base of the glacial drift. Manganiferous iron ore has been shown by drilling to extend to greater depths than this locally, but by far the larger number of deposits will probably be found to have their principal concentration above this level.

In some mines ore-bearing zones of considerable width are found in which, however, the ore itself occurs only as thin beds or lenses or in irregular scattered masses separated from each other by lean manganese iron-bearing rock. Such beds or masses of ore range from a few feet to 40 or 50 ft. in thickness and have varying lengths and depths. In other mines the ore-bearing zones contain ore scattered throughout in small or large masses in a matrix of manganese and iron-bearing rock or clay. In such cases practically the entire deposit is mined, yielding a low-grade ore. However, even in these deposits certain portions are generally found to be richer than others, some parts being so lean that it does not pay to mine them.

In tonnage the manganese iron-ore deposits range on an average between 200,000 and 500,000 tons. Bodies containing more than 500,000 tons are exceptional, although some are known which contain 1,000,000 tons or more. Many small unworkable deposits are found. The lower limit of workability is variable, depending on a number of factors, such as grade and character of ore, shape of orebody, presence or absence of included rock masses, depth of surface drift, amount of water, etc. Where conditions are favorable, orebodies containing less than 200,000 tons can be worked. Many small bodies of manganese iron ore occurring in iron-ore deposits are mined in connection with the iron ore, whereas if such bodies were surrounded by lean rock they would be unworkable. As compared with the iron-ore deposits of the district which range in size up to 10,000,000 tons and over, the manganese iron-ore bodies are small. However, on account of the greater value of the ore, many small bodies which could not be mined if they consisted of iron ore have become commercially available.

*Associated Rocks.*—The rocks found in association with manganese iron ores are mainly those characteristic of the iron-bearing formations, including ferruginous chert, ferruginous slate, cherty and slaty iron carbonate, amphibole-magnetite rock and iron ore. Green laminated slate and black graphitic slate are also commonly associated with manganese iron ore, but green chloritic schist, which is common near many of the iron-ore deposits, has not been found in association with manganese iron ore.

Graphitic slate and green slate generally form the country rock, the manganese iron-bearing formation being interlayered with them. In several localities in the northern part of the district graphitic slate has been found on one side of an ore-bearing layer and green slate on the other side. Within the manganese iron-bearing layers and composing the major part of them are the other rocks mentioned above. The manganese iron-ore bodies form but a small part of the ore-bearing layers. Probably the most common of the associated rocks is ferruginous chert, but ferruginous slate also occurs abundantly. Ferruginous chert and

ferruginous slate usually form the wall rocks of the orebodies, but in many places they are also irregularly intermixed with the ore or occur in the orebodies as horses. Iron ore locally borders manganiferous iron-ore deposits, but more commonly it is found within them as lenses or irregular masses.

As regards their occurrence, the manganiferous iron-ore deposits of the district can be roughly separated into two groups: (1) those which are associated with ferruginous chert, and (2) those which are associated with ferruginous slate. The deposits of each class possess certain characteristics which distinguish them from the other class. In deposits associated with ferruginous chert, the ore is usually found along definite horizons which were probably originally occupied by layers of more easily replaceable rock, while in those occurring in slate the ore is generally more or less scattered throughout a wide zone. The ore associated with ferruginous slate is commonly amorphous, that in ferruginous chert is in many places finely crystalline and locally shows pure manganese minerals such as pyrolusite, psilomelane, and manganite in geodes and veins. Manganite veins, however, are also common in ores in ferruginous slate. Another fairly general distinction between the two classes is that the ore occurring in association with ferruginous slate is higher in phosphorus and lower in silica than ore found in ferruginous chert.

The ferruginous chert and slate associated with manganiferous iron ore may carry varying amounts of manganese. Locally they contain less than 1 per cent. and from this there are all gradations to ore carrying 25 to 30 per cent. manganese. In iron content the rocks also show great variation ranging from 15 or 20 per cent. to 45 or 50 per cent. Besides the iron and manganese, ferruginous chert contains mainly silica, while ferruginous slate contains chiefly silica and alumina.

Cherty and slaty iron-carbonate rocks are encountered below the ore deposits at varying depths. Also, they are sometimes found bordering orebodies or occurring within them. The iron-carbonate rocks associated with the manganiferous iron ores show a considerable range in composition and texture but nearly all of them are conspicuously laminated. In places they are very cherty and consist of interlayered chert, iron and manganese carbonates and some silicate minerals while elsewhere they are predominantly slaty, containing layers of argillaceous material, carbonates, and silicates, and but little chert. Magnetite is in places an abundant constituent. Locally scattered crystals of arsenopyrite have been found.<sup>5</sup>

It is noteworthy that much of the ferrous carbonate rock occurring in association with manganiferous iron ore contains considerable man-

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<sup>5</sup> E. Newton: School of Mines Experiment Station, University of Minnesota, personal communication.

ganese carbonate. In general the manganese carbonate is not visible to the naked eye, being indistinguishable from the other carbonate minerals present. Frequently, however, one obtains specimens in which certain layers have a distinct pink tinge. It seems probable that in most cases iron, manganese, and perhaps other constituents are present in the form of mixed carbonates such as manganosiderite or ankerite. But associated with these there are doubtless also siderite, rhodochrosite, calcite, and other carbonates. Manganiferous iron-carbonate rocks may contain as much as 8 or 10 per cent.<sup>6</sup> manganese, but usually it averages less than 3 or 4 per cent.

Amphibole-magnetite rock and magnetitic slate are in many places associated with cherty and slaty iron-carbonate rocks, in fact most of the latter contain a varying amount of magnetite and silicate minerals. This indicates that very severe metamorphism accompanied the folding of the rocks. Intrusive igneous rocks which might have effected this metamorphism have not been found in the vicinity of manganiferous iron-ore deposits.

*Character and Grade.*—The manganiferous iron ore presents a variety of different phases depending more or less upon the nature of the rock with which it is associated. Some ore is black, some is yellowish, reddish or brownish-black. Some of it consists of fairly pure manganese oxides coarsely and irregularly intermixed with hydrated ferric oxides, giving mottled black and yellow, black and brown, or black and red ore, the color depending upon the degree of hydration of the ferric oxides. In places the intermixing is so coarse that pieces of high-grade manganese ore of considerable size can be extracted from the mixture, leaving a low-grade manganiferous iron ore. The iron and manganese oxides are generally amorphous and may be soft and porous or hard and massive. Locally, however, veins and geodes of various crystalline oxides are found. Other and more common phases of the manganiferous iron ore consist of a mixture of iron and manganese oxides so intimate that it is impossible to separate them mechanically. Such ores are usually uniformly black, reddish-black, or brownish-black and amorphous, the separate oxides composing them being indistinguishable. Between these two kinds of ore there are all gradations, manganiferous iron ore low in manganese being intermixed in all proportions with manganiferous iron ore high in manganese.

In still other phases of the ore, generally of lower grade, manganese ore instead of being mixed with iron ore may be intermixed with cherty or argillaceous iron-bearing or manganiferous iron-bearing rock. Thus there are manganese ores or manganiferous iron ores mixed with chert, slate, or clay which may carry varying amounts of manganese and iron.

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<sup>6</sup> E. Newton: personal communication.



Such ores are generally high in silica or alumina and with increasing amounts of these substances pass into manganiferous iron-bearing rock.

Both hard and soft ore occur in the deposits. The hard or soft character is not indicative of difference in grade. Some high-grade ores are hard, others are soft. Masses of almost pure manganese oxide may be quite soft or fairly hard, depending on the nature of the minerals composing them. Low-grade manganiferous iron ores also vary in hardness. It may be said in general that most of the hard ores are associated with cherty manganese-bearing and iron-bearing rocks and most of the soft ores with slaty or clayey manganese- and iron-bearing rocks.

The manganese minerals found in the Cuyuna manganiferous iron ores are manganite, psilomelane, wad, and pyrolusite, while the iron minerals are hematite and hydrated ferric oxides of the goethite-limonite series. As a general rule these various minerals cannot be distinguished in the ores, but locally where the separate minerals are distinguishable all of them have been found. In some mines, as the Algoma, veins and geodes of pure crystalline pyrolusite occur. Elsewhere veins of crystalline manganite or geodes consisting of successive layers of crystalline limonite or goethite with psilomelane and manganite, or psilomelane and pyrolusite are found. The order of formation commonly is: limonite-goethite, psilomelane, manganite, and pyrolusite. Where manganite is present pyrolusite usually is not found, the pyrolusite probably being an alteration product of manganite.

Besides the oxides there are present in manganiferous iron-ore deposits varying quantities of other minerals, both ore and gangue. Among the ore minerals are the carbonates, rhodochrosite and siderite or the mixed carbonates of iron, manganese, calcium, and magnesium such as ankerite, or of iron and manganese such as manganosiderite. These minerals are often present in appreciable amounts and in some mines form a considerable percentage of the ore. They are usually pinkish or yellowish-brown, finely crystalline, and are intermixed with more or less iron and manganese oxide. In most places the carbonates are probably partly altered phases of the original cherty or slaty manganiferous iron-carbonate rock, but locally they are formed by later impregnation, as veins of these minerals cut the oxidized iron-bearing formation.

The principal gangue material associated with the manganiferous iron ore is country rock in which the ore is concentrated. This is mainly ferruginous and manganiferous chert, red slate, and yellow clay. In depth unaltered phases of the iron and manganese-bearing rock, such as cherty and slaty carbonate and green slate, become common. Besides the rock impurities, however, certain gangue minerals form veins and local deposits in the ore. The principal one of these minerals is quartz, which is commonly found in veins alone or associated with manganite, psilomelane, ferric hydroxides or manganese and iron carbonates. In

some mines quartz is so abundant as to materially reduce the grade of the ore. Other gangue minerals found are calcite, iron silicates and in a few places barite. In one or two of the mines there are masses of fine-grained epidote the origin of which is uncertain. It is possible that among the gangue minerals there are also manganese silicates such as rhodonite associated with the iron silicates or with rhodochrosite. Such silicates, however, have not been definitely recognized.

As may be judged from the description of the varying character of the ore, there is also a great variation in the grade. This is true not only of different deposits but within short distances in the same deposit, especially across the strike of the formation. In large part these variations are due to original differences in the character of the beds in which the manganese and iron ore are now concentrated. There are also variations along the strike of the beds, however, which are not attributable to original differences of material, but rather to irregularities in ore concentration. Thus lenses of manganiferous iron ore may along the strike run into high-grade iron ore or irregular masses of iron ore may be inclosed within the manganiferous iron-ore deposits. In the same manner, masses of lean or barren rock may be inclosed with the orebodies. The result of this is that the manganese content of the orebodies may vary from 2 or 3 per cent. to 15 or 20 per cent. within very short distances. The iron content varies similarly, but not so irregularly nor usually through so great a range in short distances. Silica and alumina are fairly constant in quantity along the same beds, but vary in amount in different beds according as the beds are cherty or argillaceous.

In view of these irregularities in the composition of the ore there is considerable difficulty in maintaining a uniform product from the mines. This difficulty is overcome, however, by mixing ores from different portions of the orebody and also by marketing several distinct grades of ore. Some of the mines offer as many as four different grades of manganiferous iron ore.

Iron and manganese oxides with silica and alumina form the principal constituents of the ore and in most cases their percentages, together with that of water of hydration, add up to approximately 100. In places lime and magnesia are present in considerable amount, but they are more or less local in their occurrence. In some ores, the amount of volatile material present is abnormally high, due in part to the high moisture content, and in part to the presence of carbonates. The iron content of the manganiferous iron ores usually ranges between 25 and 50 per cent. while the manganese content ranges up to 30 or 35 per cent. Silica is present up to 16 or 18 per cent. and alumina varies between 1.5 and 4 per cent. The combined water in the ore usually ranges between 8 and 12 per cent.; the higher-grade ores are lower in moisture. In deposits where the iron and manganese are present in large part as carbonates, carbon dioxide is found in considerable amounts.

In general it may be said that as regards grade there are two principal classes of manganiferous iron ores in the district: (1) low-phosphorus ores, usually high in silica, and (2) high-phosphorus ores which are generally moderately low in silica. Low-phosphorus ores are commonly

TABLE 1.—*Analyses of Manganiferous Iron Ores from the North Range, Cuyuna District, Showing Range in Percentage of Different Constituents*

| Mn    | Fe    | P     | SiO <sub>2</sub> | Combined<br>Fe and Mn | Mn    | Fe    | P     | SiO <sub>2</sub> | Combined<br>Fe and Mn |
|-------|-------|-------|------------------|-----------------------|-------|-------|-------|------------------|-----------------------|
| 1.01  | 50.75 | 0.276 | .....            | 51.76                 | 18.55 | 39.78 | 0.056 | .....            | 58.33                 |
| 1.16  | 55.92 | 0.091 | 8.24             | 57.08                 | 19.40 | 35.01 | 0.128 | .....            | 54.41                 |
| 1.49  | 50.66 | 0.122 | .....            | 52.15                 | 19.76 | 35.48 | 0.041 | .....            | 55.24                 |
| 2.37  | 55.69 | 0.391 | .....            | 58.06                 | 20.53 | 32.44 | 0.076 | 9.95             | 52.97                 |
| 2.42  | 54.45 | 0.034 | .....            | 56.87                 | 20.73 | 31.25 | 0.075 | 12.94            | 51.98                 |
| 2.44  | 56.42 | 0.134 | .....            | 58.86                 | 21.70 | 33.52 | 0.027 | .....            | 55.22                 |
| 3.13  | 58.76 | 0.123 | .....            | 61.89                 | 21.75 | 30.68 | 0.060 | .....            | 52.43                 |
| 3.61  | 51.36 | 0.058 | .....            | 54.97                 | 21.80 | 30.27 | 0.211 | .....            | 52.07                 |
| 4.49  | 51.90 | 0.085 | 9.25             | 56.39                 | 22.51 | 34.06 | 0.058 | 9.94             | 56.57                 |
| 4.96  | 46.52 | 0.044 | .....            | 51.48                 | 22.63 | 34.86 | 0.105 | .....            | 57.49                 |
| 5.09  | 55.58 | 0.023 | .....            | 60.67                 | 23.45 | 27.27 | 0.059 | .....            | 50.72                 |
| 5.16  | 50.64 | 0.107 | .....            | 55.80                 | 23.63 | 35.38 | 0.062 | 9.69             | 59.01                 |
| 5.90  | 49.75 | 0.324 | .....            | 55.65                 | 23.70 | 30.66 | 0.258 | .....            | 54.36                 |
| 6.75  | 54.53 | 0.021 | .....            | 60.28                 | 24.12 | 34.33 | 0.087 | .....            | 58.45                 |
| 6.76  | 47.58 | ..... | 11.65            | 54.34                 | 24.32 | 33.87 | 0.240 | .....            | 58.19                 |
| 6.81  | 45.22 | 0.173 | .....            | 52.03                 | 24.36 | 30.85 | 0.066 | .....            | 55.21                 |
| 7.63  | 47.26 | 0.139 | .....            | 54.89                 | 25.00 | 30.61 | 0.076 | .....            | 55.61                 |
| 7.98  | 47.36 | ..... | 13.97            | 55.34                 | 25.19 | 30.96 | 0.141 | .....            | 56.15                 |
| 8.02  | 53.93 | 0.029 | .....            | 61.95                 | 25.59 | 31.05 | 0.333 | .....            | 56.64                 |
| 8.05  | 52.59 | 0.212 | .....            | 60.64                 | 26.20 | 31.19 | 0.080 | .....            | 57.39                 |
| 9.18  | 44.03 | 0.028 | .....            | 53.21                 | 26.21 | 31.20 | 0.294 | .....            | 57.41                 |
| 9.45  | 48.37 | 0.076 | .....            | 57.82                 | 26.62 | 36.55 | 0.051 | .....            | 63.17                 |
| 9.48  | 47.03 | 0.325 | 3.18             | 56.51                 | 27.15 | 31.91 | 0.064 | .....            | 59.06                 |
| 10.72 | 44.07 | ..... | 12.39            | 54.79                 | 27.52 | 28.27 | 0.156 | .....            | 55.79                 |
| 10.84 | 39.38 | 0.061 | .....            | 50.22                 | 27.88 | 23.20 | ..... | 16.22            | 51.08                 |
| 11.08 | 44.41 | 0.021 | .....            | 55.49                 | 28.11 | 27.94 | 0.172 | .....            | 56.05                 |
| 11.61 | 45.90 | 0.152 | .....            | 57.41                 | 28.75 | 31.56 | 0.083 | .....            | 60.31                 |
| 12.14 | 41.07 | 0.138 | .....            | 53.21                 | 29.51 | 22.03 | 0.121 | .....            | 51.54                 |
| 12.31 | 41.96 | 0.036 | .....            | 54.27                 | 29.92 | 22.71 | 0.073 | .....            | 52.63                 |
| 13.91 | 38.05 | 0.050 | .....            | 51.96                 | 30.10 | 28.89 | 0.041 | .....            | 58.99                 |
| 13.95 | 40.90 | 0.023 | .....            | 54.85                 | 31.46 | 28.22 | 0.043 | 6.72             | 59.68                 |
| 14.65 | 40.56 | 0.045 | .....            | 55.21                 | 31.50 | 32.42 | 0.063 | .....            | 63.92                 |
| 14.90 | 38.48 | 0.279 | .....            | 53.38                 | 32.40 | 31.24 | 0.038 | .....            | 63.64                 |
| 14.91 | 38.20 | 0.307 | .....            | 53.11                 | 33.28 | 25.48 | 0.034 | 8.85             | 58.76                 |
| 15.06 | 38.79 | 0.046 | .....            | 53.85                 | 33.44 | 18.08 | 0.088 | .....            | 51.52                 |
| 15.37 | 34.74 | 0.062 | 15.10            | 50.11                 | 33.66 | 27.36 | 0.129 | .....            | 61.02                 |
| 16.30 | 35.00 | 0.140 | .....            | 51.30                 | 34.10 | 22.34 | 0.046 | .....            | 56.44                 |
| 16.96 | 38.67 | 0.088 | .....            | 55.63                 | 34.83 | 24.69 | 0.059 | 9.29             | 59.52                 |
| 17.18 | 33.71 | 0.041 | 13.44            | 50.89                 | 35.85 | 24.96 | 0.043 | .....            | 60.81                 |
| 17.55 | 39.25 | 0.255 | .....            | 56.80                 | 36.90 | 17.72 | 0.032 | .....            | 54.62                 |
| 18.00 | 32.37 | 0.198 | 13.60            | 50.37                 | 37.94 | 19.93 | 0.071 | .....            | 57.87                 |
| 18.00 | 37.03 | 0.085 | 12.72            | 55.03                 |       |       |       |                  |                       |

associated with cherty iron-bearing formation and high-phosphorus ores with slaty iron-bearing formation. Because of their association with different kinds of rocks, high-phosphorus orebodies and low-phosphorus orebodies generally occur separated from each other. This association, however, does not hold universally, and in some places high- and low-phosphorus ores occur together in the same series of rocks or even in the same deposit. Some mines produce only one class of ore, while in other mines orebodies of both classes are found closely associated.

In Table 1 are grouped a large number of analyses of manganiferous iron ores taken from practically all the known manganiferous iron-ore deposits in the north range. Many of them represent samples taken in drilling while many of them are of samples taken in mines. They were selected to show the general range in iron, manganese, and phosphorus content and in some cases the silica content as well. In most instances, in order to show the range of these constituents, several analyses are given of ores which have the same manganese content but which show differences in the percentage of iron and phosphorus. Only ores in which the combined iron and manganese content is more than 50 per cent. are included in the table, although much ore is being shipped at the present time in which the combined metallic content is considerably less.

The analyses show the great variation in the manganese content of the ores. They show also that ore containing a given percentage of manganese may vary greatly in the amount of the other constituents present, especially the phosphorus. Aside from minor irregularities, the iron content decreases with increase of manganese content, while the silica decreases with increase in content of combined metals, and *vice versa*. The phosphorus content shows no such regularity, the variations being independent of the iron and manganese content. In general, however, it may be said that the ores which are highest in manganese are usually of the class of low-phosphorus ores.

*Origin.*—The genesis of the manganiferous iron ore of the Cuyuna district is in general similar to that of the iron ore. There are, however, certain minor differences due to the different behavior of iron and manganese under oxidizing conditions. In order to bring out these differences more clearly, it may be well to outline briefly the principal steps in the formation of the iron ore.

1. The original iron-bearing solutions, the source of which may be disregarded in this discussion, deposited their iron under reducing conditions mainly as ferrous carbonate but perhaps in part as ferrous silicate. During the precipitation of the iron salts, colloidal silica and clay carried by the water were deposited also. Sometimes the iron salts were deposited in greater quantity and sometimes the clay or silica, thus giving rise to successive laminæ of these various materials. Fine laminæ of

silica or of clay became interlayered with laminae of iron salts. The general character of the sediments varied according to the material predominating in the deposits. When clay was in excess a sediment was formed which ultimately gave rise to green slate or slaty iron carbonate, and when colloidal silica was in excess a sediment which developed into cherty iron carbonate was formed. However, these different types occur interlaminated or interbedded.

It seems probable that not all the iron was deposited in the ferrous form but that locally oxidizing conditions prevailed and iron was deposited as ferric hydroxide. As in case of the iron-carbonate deposition, clay and colloidal silica were probably present and were deposited in greater or less amounts. Upon consolidation, these sediments gave rise to ferruginous slate and ferruginous chert with perhaps some beds of fairly pure hematite or hydrated hematite. Such rocks would be similar in general to the rocks formed by the surface oxidation and weathering of cherty and slaty iron-carbonate rocks. It is believed, however, from the evidence afforded in various parts of the district, that the iron deposited in the ferric form originally was very small in amount.

2. Consolidation of the sediments took place and igneous rocks became associated with them mainly as local intrusive sheets and sills, but perhaps in part also as extrusives.

3. All the rocks were profoundly folded and metamorphosed. The argillaceous rocks and many of the igneous rocks were rendered schistose, producing green slate, chloritic schist, and other associated rocks. The iron-carbonate rock was recrystallized and much of it was altered to amphibole-magnetite rock and magnetitic slate. Where original beds of ferruginous chert were present they were altered to jaspilite. The layers of oolitic cherty ferric oxide in the jaspilite were probably original and were not formed by the alteration of ferrous silicate (greenalite) as is the case with similar material known as taconite found in the Mesabi district. This is indicated by the fact that apparently there was no intervening erosion period during which the alteration of the greenalite could have taken place before the metamorphism to jaspilite.

4. Later intrusions of igneous rocks occurred locally which probably caused further metamorphic changes.

5. The rocks were elevated and eroded, the upper portions being exposed to processes of weathering and oxidation. The cherty and slaty ferrous carbonate rocks were altered to hematitic and limonitic chert and hematitic and limonitic slate for varying distances from the surface. The green slate and chloritic schist associated with the iron-bearing rocks were altered to ferruginous slate and hematitic schist respectively. However, oxidation was not the only result of surface action. Meteoric waters percolating through the rocks dissolved and carried away silica, leaving the iron oxide more concentrated. In much of the ferruginous

chert in which the iron content was originally fairly high and which was favorably situated with reference to underground circulation, this process resulted in the formation of iron-ore deposits. Ferruginous slates, on the other hand, on account of their high alumina content and imperviousness to water, do not give rise to iron ores. The iron-ore deposits, however, were probably not formed exclusively by the removal of silica. There is considerable evidence to show that to some extent there has also taken place a movement of the iron. Ferric oxide is in many places found as veins cutting across beds of iron-bearing rock, as irregular impregnations in rocks which originally were free from ferric oxide or contained but

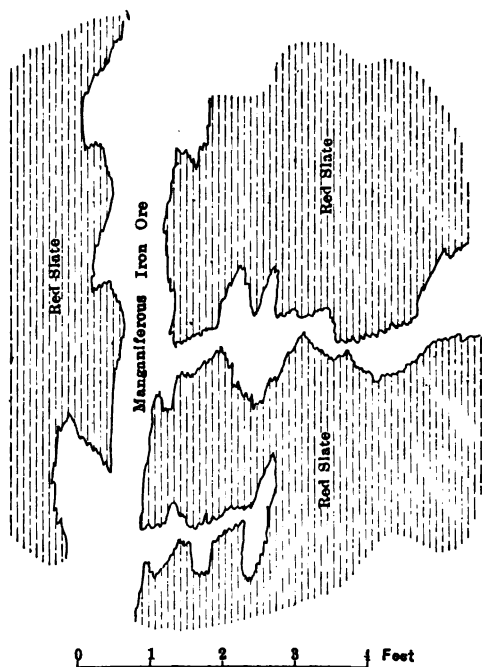


FIG. 4.—SKETCH SHOWING THE REPLACEMENT OF FERRUGINOUS SLATE BY MANGANESE OXIDE ALONG BEDDING PLANES AND ALONG FRACTURE PLANES AT RIGHT ANGLES TO THE BEDDING PLANES.

small amounts, and in the crystalline form in geodes. While all these phenomena show that movement of iron has taken place, there is little doubt that the principal concentration in the formation of iron-ore deposits resulted from the removal of silica.

In the preceding paragraphs are stated briefly the various steps which have led to the formation of the iron-ore deposits. The manganiferous iron ores have passed through practically the same stages in their formation as the iron ores. The original cherty and slaty iron-carbonate rocks which have been encountered locally in the manganiferous iron-bearing

beds in association with the ores, nearly always show small and varying percentages of manganese present in the form of rhodochrosite (manganese carbonate) associated with siderite or in the form of the double carbonate of iron and manganese, manganosiderite. These manganiferous iron-carbonate rocks have undergone the same changes that have been described for the iron-carbonate rocks. They have been dynamically metamorphosed, resulting in recrystallization and in the development locally of considerable magnetite and amphibole. Upon exposure to processes of weathering and oxidation they were altered to manganiferous ferruginous chert, manganiferous ferruginous slate, and manganiferous iron ore.

While the formation of the manganiferous iron-ore deposits has in general been similar to that of the iron-ore deposits, there are certain differences and these have resulted in differences in the nature of the two classes of deposits. The iron-ore deposits, as stated, were formed mainly by the removal of silica from ferruginous chert rich in iron. Some solution and redeposition of iron doubtless took place during its formation, but it was of minor importance. In the formation of manganiferous iron-ore deposits, on the other hand, the solution and redeposition of manganese has played a very important part, although in this case the removal of silica also has been a factor in the ore concentration. As a result of the predominance of solution and redeposition, the manganiferous iron-ore deposits have been formed not in ferruginous chert alone, but also in ferruginous slate. Many important deposits occur in ferruginous oxide along fracture planes, bedding planes, cleavage planes, and other water channels, or as irregular nodular impregnations throughout the mass of the rock. While iron-ore deposits as a rule form large bodies containing locally large and small masses of residual rock, the manganiferous iron-ore deposits are more largely rock, *i.e.*, ferruginous chert, slate and clay, in which the ore occurs as large and small irregular impregnations, nodules, veins, geodes, and replacements. The proportion of ore to rock varies greatly in manganiferous iron-ore deposits; locally ore masses are practically continuous with one another while elsewhere they are scattered. In general there is much more rock in manganiferous iron-ore deposits than in iron-ore deposits.

The principal evidence of the importance of solution and redeposition in the formation of manganiferous iron-ore deposits is: (1) occurrence of veins of manganese minerals and impregnation of the adjacent rock by manganese oxide for varying distances from the veins; (2) similar replacement of rock by manganese oxide on both sides of fracture planes, bedding planes, or other openings allowing the passage of water (Fig. 4); (3) the gradual decrease in manganese content of such replaced material away from the fissures; (4) the abundant occurrence of geodes and veins containing crystalline manganese oxide; (5) the occurrence of concentric

layers of different manganese minerals in such crystalline material; (6) the prevalence of nodular and concretionary forms in the ore deposits; and (7) the close association of the manganese oxides with vein quartz.

Just how important the removal of silica has been in the formation of manganiferous iron-ore deposits is uncertain. Doubtless its importance is considerable in those deposits which are found in association with manganiferous and ferruginous chert, but even in these, evidence of solution and redeposition of manganese exist. In the deposits occurring in ferruginous slate or clay, the leaching of silica has probably not been a very important factor in the concentration of the ore.

It appears, therefore, that the same processes have been active in the formation of both iron-ore and manganiferous iron-ore deposits in the Cuyuna district. The two metals, however, have been affected somewhat differently. On account of the greater solubility of manganese, its distribution through the rocks has been greater. It has travelled farther and is not so closely confined to the layers that contained it originally. However, the solubility is apparently only slightly in excess of that of iron, and for this reason the two metals in general tend to be concentrated together. As already stated, there is also a small amount of iron dissolved and reprecipitated. The differences noted, therefore, are differences of degree and not of kind.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representatives of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Some Unusual Features in the Microstructure of Wrought Iron

BY HENRY S. RAWDON,\* WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

THE structure of wrought iron as usually described by metallographists and workers in metal in general is that of a fairly pure iron. Impurities, if present, are usually considered as being in solid solution in the crystals of the ferrite matrix or as forming part of the ever present "slag streaks." Attention is herein directed to one type of these dissolved impurities; to the detection of such impurities, particularly in low-grade irons, and to their possible influence on the physical properties of the metal. The attention of the Bureau of Standards was first directed to wrought iron of the peculiar and unusual characteristics to be described later, in material which had failed in service and was submitted for test. The examination was extended to other grades of wrought iron, to see whether such features are of common occurrence in this class of material.

### MICROSTRUCTURE OF WROUGHT IRON

#### *Usual Structure*

The general microstructure of wrought iron is so well known that a detailed description of it here is needless. There is shown in Fig. 2 the structure of a sample of Swedish iron which illustrates well the matrix of ferrite crystals in which are embedded the slag threads so characteristic of the puddling process and the subsequent working of the material. The usual commercial product contains varying amounts of the impurities commonly associated with iron and steel, *i.e.*, manganese, phosphorus, silicon, and sulphur together with small amounts of carbon. Of these, the larger part of the phosphorus and some of the silicon present are held in solid solution in the ferrite while most of the remaining impurities, other than carbon, exist in the inclosures of slag. Aside from the discontinuities introduced by the slag streaks the ferrite matrix has the microstructure and appearance of a pure metal. No definite orientation of the various crystals or grains is apparent; deep etching of the specimens reveals no intracrystalline features (*i.e.*, within the grains) other than those noted in "pure" metals in general (etching pits).

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*Unusual Features*

The unusual features noted may be best illustrated by a brief description of the structure of the specimen in which they were first observed in abundance. This piece was a wrought iron eye-bar, a tension member of a railway bridge which after about 30 years' service was modified to suit the increased traffic. After this modification, the member referred to above failed. The microscopic examination of the metal

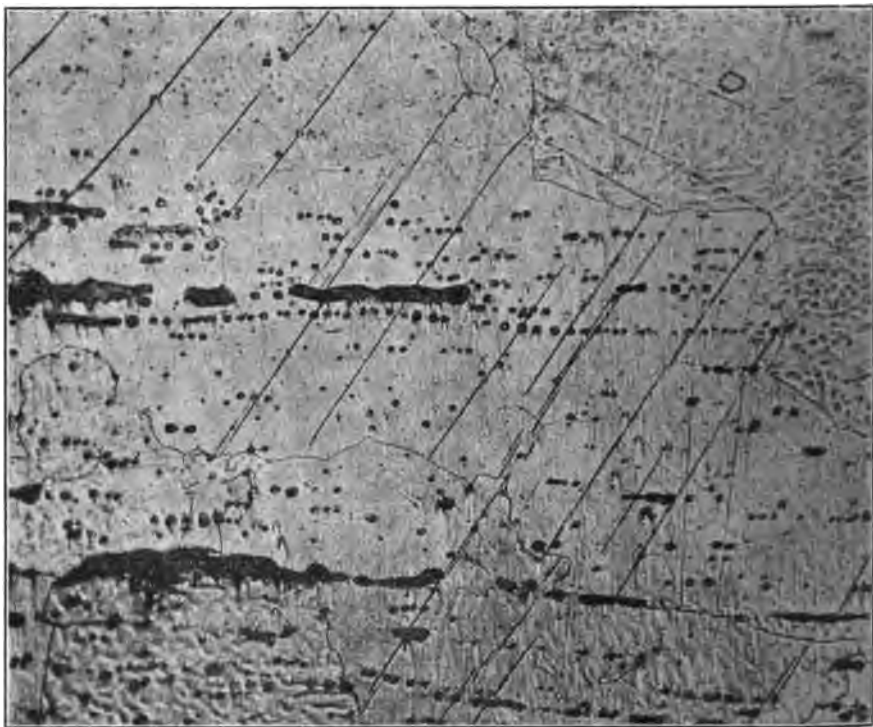


FIG. 1.—THE DARK LINES, PARALLEL WITHIN ANY SINGLE GRAIN, ARE THE NEUMANN LINES. THE MOTTLED APPEARANCE OF THE CORNER GRAINS INDICATES THE NON-HOMOGENEITY OF THE FERRITE DUE TO THE UNEQUAL DISTRIBUTION OF THE PHOSPHORUS. SPECIMEN: W1 (TABLE 1), LONGITUDINAL SECTION. ETCHING, 10 PER CENT. ALCOHOLIC NITRIC ACID.  $\times 100$ .

revealed, in addition to the usual structure of wrought iron, several structural features so striking as to mark the sample at once as a wrought iron of very unusual properties and composition.

The ferrite crystals presented a peculiar mottled appearance, particularly after prolonged etching with an acid reagent. This etch pattern was not found over the entire surface of the specimen but was restricted to certain streaks throughout the metal. Particularly was it found associated with crystals unusually large in size (Fig. 5). A series of

measurements upon those portions of the material in which such etch patterns were developed in abundance showed the crystals to be as large as 0.25 by 2.00 mm.; a similar set of measurements upon the crystals of the material of Fig. 2 showed the average size to be 0.144 by 0.133 mm. This latter set may be taken as quite characteristic of the grain size of good grades of wrought iron.

The etch pattern referred to above is illustrated in Figs. 1, 3, 4, 5. By using the copper chloride etching reagent referred to later, these patterns may be developed in a very striking manner. They sometimes consist of broad parallel bands, as seen in cross-section, extending across the elongated crystals (Fig. 3) and bearing some resemblance to twinned



FIG. 2.—MICROSTRUCTURE OF A LONGITUDINAL SECTION OF SWEDISH IRON. ETCHING, 2 PER CENT. NITRIC ACID.  $\times 100$ .

crystals. More often they present an indefinite mottled appearance of light and dark areas very similar to the shadows cast by the sunlight streaming through between the leaves of a tree (Figs. 1 and 5). The crystals of ordinary wrought iron will not exhibit such etch patterns even after very prolonged etching.

The examination of the metal close up to the fracture, which occurred during the service of the material, shows that the break occurred through the crystals and parallel to the markings constituting the mottled etch-pattern at that point (Fig. 14). Many of the crystals close to the face of the fracture show another variety of intracrystalline markings. On casual examination these may be mistaken for scratches left by poor

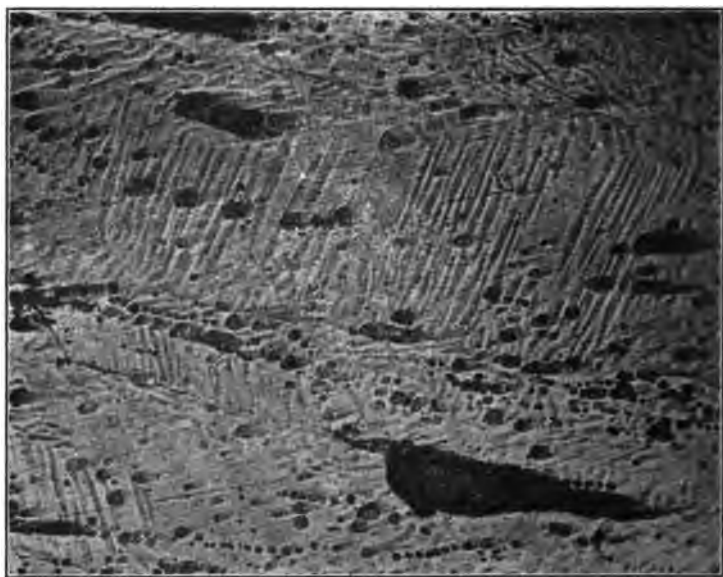


FIG. 3.—SPECIMEN SHOWING THE MOTTLLED ETCH PATTERN OCCURRING AS PARALLEL TRANSVERSE BANDS WITHIN THE GRAINS. ETCHING, 10 PER CENT. NITRIC ACID IN ALCOHOL.  $\times 100$ .

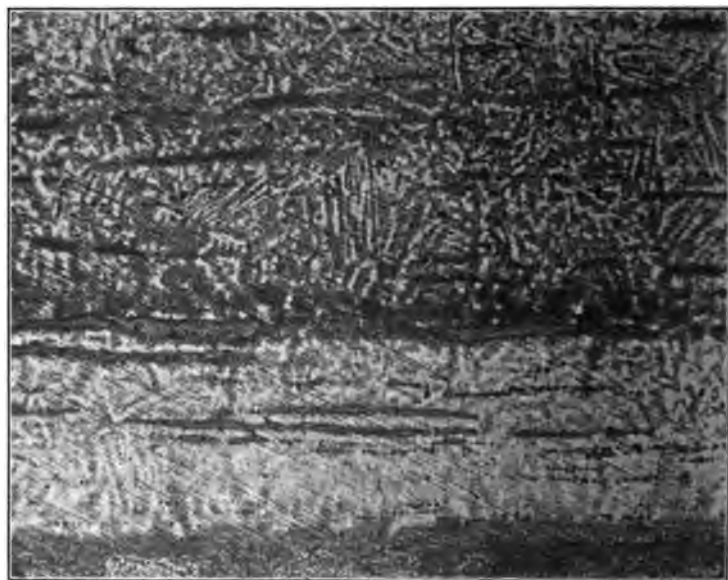


FIG. 4.—WROUGHT IRON PIPE, LONGITUDINAL SECTION. ETCHING, ALCOHOLIC COPPER CHLORIDE ACIDULATED WITH HYDROCHLORIC ACID. (STEAD'S REAGENT.) THE PORTIONS SHOWN ARE FROM THE STREAKS UPON WHICH COPPER DID NOT DEPOSIT.  $\times 100$ .

polishing of the material (Fig. 1). Closer inspection, however, shows that these markings are parallel to one another within any one crystal and terminate very abruptly at the crystal boundaries. If a slag enclosure lies in the course of one of these markings, the line ends abruptly, reappearing on the other side of the "slag," thus indicating clearly that it is not a scratch. In some crystals, a second set usually not so well developed as the primary ones is seen; the lines comprising this second set are parallel to one another but form a definite angle with those of the first. Crystals showing markings of this general character in considerable numbers usually present a smoother surface and show less of the shadow-



FIG. 5.—THE STREAKS EXHIBITING THE UNUSUAL ETCH PATTERN ARE VERY OFTEN COMPOSED OF GRAINS THAT ARE MUCH LARGER THAN THE AVERAGE FOR THE PIECE. ETCHING, 10 PER CENT. NITRIC ACID. SAME MATERIAL AS FIG. 3.  $\times 100$ .

like etch-pattern described above than do those crystals in which markings of this class do not occur.

The obvious conclusion is that the non-homogeneity of the individual crystals as indicated by the mottled etch-pattern is to be attributed to some impurity dissolved in the iron but not uniformly diffused throughout the crystal. Robin<sup>1</sup> has called attention to the fact that in ferrite containing considerable phosphorus, *e.g.*, 1 per cent. such a non-homogeneity may exist. Stead,<sup>2</sup> in his extensive studies of the segregation of

<sup>1</sup> F. Robin: *Traité de Metallographie* (Paris, 1912), 184.

<sup>2</sup> *Journal of the Iron & Steel Institute* (1915), 91, 140. Stead here summarizes his previous work and gives a quite complete bibliography of the literature on phosphorus in iron.

phosphorus in iron and steel, makes but very slight reference to this type of intracrystalline variation of structure which Robin attributes to the non-uniform distribution of phosphorus in the ferrite which forms the body of the crystal. Saklatwalla<sup>3</sup> in his work on the form in which phosphorus occurs in iron, says that up to 1.4 per cent. of phosphorus, a single constituent consisting of the solid solution occurs. Konstantinow's work<sup>4</sup> deals primarily with the alloys high in phosphorus, and those low in this element receive but scant attention.

### MATERIALS EXAMINED

The variations from the usual microstructure of wrought iron, first noted in the wrought-iron bridge member which failed in service, appeared so striking and unusual in character and have been so meagerly described in the literature on the subject that the examination was continued to other grades of iron. From the accumulated series of wrought-iron specimens which have been submitted to the Bureau of Standards from time to time for examination, there were chosen some 35 samples for detailed microscopic examination. Much of this material had been submitted as "unsatisfactory" and one of the purposes of the examination was to see whether such material is characterized by microstructural features similar to those noted above. In Table 1 are listed the materials used. The examination was made by means of Stead's method of etching,<sup>5</sup> making use of an alcoholic cupric-chloride solution acidulated with hydrochloric acid, by which the segregation of phosphorus is shown by the differential precipitation of copper on the surface of the metal. The precipitation of copper on the areas relatively low in phosphorus is much heavier than on the portions of higher phosphorus content, so that the approximate distribution of this element is rendered visible to macroscopic examination. The "high" and "low" phosphorus streaks thus revealed were then examined microscopically to see if the unusual mottled etch-pattern occurred in either.

The results of the examination indicate that the unusual microstructure noted in the failed eye-bar, while not to be regarded as a common feature of wrought iron or always associated with iron of inferior grade, is not unique for the single specimen in which it was first observed. It was noted, however, that such features were found only in material, or portions of material, that, according to Stead,<sup>6</sup> are to be regarded as relatively high in phosphorus.

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<sup>3</sup> *Metallurgie* (1908), 5, 336, 711.

<sup>4</sup> *Zeitschrift für anorganische Chemie* (1910), 66, 209.

<sup>5</sup> *Loc. cit.*

<sup>6</sup> *Loc. cit.*

## NATURE OF THE UNUSUAL FEATURES OBSERVED

*Comparison with Iron-Phosphorus Alloys*

For purposes of comparison, to make certain that the unusual microstructure observed is indicative of a non-uniform distribution of phosphorus, a series of iron-phosphorus alloys was prepared. The phosphorus content from 0.014 to 0.48 per cent. for the series of six. Electrolytic iron<sup>7</sup> which had been melted *in vacuo* was used.

Fig. 7 shows the microstructure of an iron-phosphorus alloy thus prepared containing 0.37 per cent. phosphorus. The metal in solidifying,

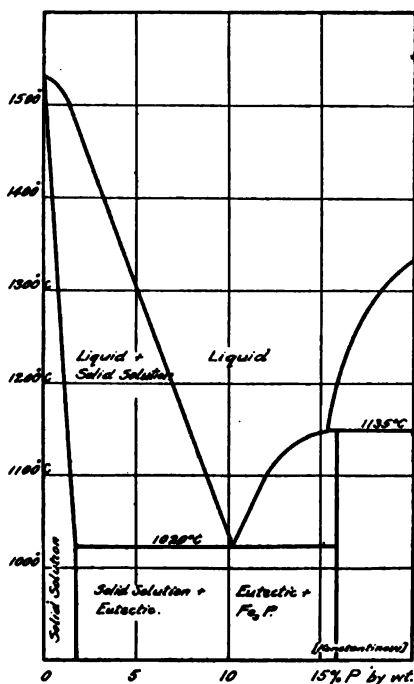


FIG. 6.—PORTION OF THE CONSTITUTIONAL DIAGRAM OF THE IRON-PHOSPHORUS ALLOYS.

freezes selectively, *i.e.*, the solid solution which separates first from the melt is relatively low in phosphorus and the portions successively added are progressively richer in this element. A eutectic consisting of the solution of phosphorus in iron and iron phosphide, analogous to pearlite in steel, formed out of the portion which solidified last of all.

The alloys containing amounts of phosphorus lower than 0.37 per cent. showed the same general structure as Fig. 8, minus the eutectic.

<sup>7</sup> Cain, Schramm, and Cleaves: *Bureau of Standards Scientific Paper No. 266*. Iron, prepared as here described, was used.

The equilibrium diagram based on Konstantinow's work<sup>8</sup> is shown in Fig. 6. According to this, phosphorus is soluble in iron up to about 1.7 per cent., when the system is in equilibrium. Two of the alloys as cast, containing 0.37 and 0.48 per cent. phosphorus respectively, though far below the limit of solubility, show the heterogeneous character of those above this limit. This heterogeneity of structure in low-phosphorus alloys has not been emphasized by Saklatwalla or Konstantinow in their work on this series. The very slow diffusion of phosphorus in the ferrite is the cause of the heterogeneous structure, a condition very apt to prevail in practice, particularly if the heating and working of the material subsequent to the initial rolling is slighted. On annealing the alloy, the eutectic disappears by the phosphide passing into solution in the ferrite and, if the heating is continued, the crystal becomes homogeneous by the diffusion of the phosphorus in solution. Upon etching no pronounced mottled etch-pattern now appears, only a coloring, yellow or brown, of the surface is seen.

The comparison of the structure of the iron-phosphorus alloys with that observed in the wrought irons examined is instructive and very suggestive as to the nature of the markings there seen. The brown or dark bands and spots forming the mottled etch-patterns are the portions of the ferrite rich in phosphorus and each individual crystal in the streaks of the metal showing such mottled appearance is to be regarded not as a simple entity but as a rather complex aggregate. That the different portions of a single ferrite crystal in such portions vary considerably in their properties is to be inferred from the marked variation in composition.

#### *Chemical Composition of Materials Showing Such Features*

The analyses for phosphorus of many of the materials listed (see Table 1) show that, though such unusual features of structure as have been described are invariably associated with irons which are rather high in phosphorus, one cannot predict with certainty their presence from a knowledge of the average phosphorus content alone. Some of the samples, *e.g.*, W 25, 31, 32, and 33 though comparable with W 11, 12, or 16 in respect to the phosphorus content, showed no traces of these unusual features.

#### *Etching of Samples*

Upon continued polishing of the specimens before etching, a faint trace of the markings may be seen and recognized by one after becoming familiar with this type of iron. By acid etching, preferably using a 5 or even a 10 per cent. alcoholic solution of nitric acid, the etch-pattern may be satisfactorily developed. The specimen shows to the eye the brown

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<sup>8</sup> *Loc. cit.*



and purple oxide tints similar to those which are often observed when hardened and tempered steels are etched with an acid reagent.

By using the cupric chloride solution described by Stead the heterogeneous structure may be developed much more strikingly than with acid alone. It was found, however, that the very definite procedure as described by Stead is not necessary; the sample may be immersed in an excess of the reagent as is the usual practice in etching. Ordinarily, when wrought iron is etched with this copper chloride reagent, it shows the presence of the streaks rich in phosphorus by the deposition of copper on the purer portions so that the "segregation streaks" appear more lightly colored in contrast to the matrix. On wrought irons high in phosphorus throughout and in the iron-phosphorus alloys of rather high phosphorus content, *e.g.*, 0.37 or 0.48 per cent., such a deposition of copper was not observed even after 15 to 20 min. immersion in the etching reagent. The brown tint which results in such cases is very similar in appearance to that obtained with simple acid etching, though the action is more rapid when the copper chloride is used. This coloration has been attributed by Stead to a much retarded deposition of copper. Several samples of the iron-phosphorus alloys and high-phosphorus layers cut out of wrought iron were etched to a pronounced brown by the copper-chloride reagent by immersing for 20 to 40 min. The darkened surface layer was dissolved off with nitric acid after thoroughly washing the specimen, and the resulting solution showed a faint but clearly perceptible test for copper.

Though the amount of copper precipitated is very slight, the decided increase of contrast in the etched specimen when the acid copper-chloride reagent is used over that obtained with acid alone warrants the conclusion that it is to the copper that the more rapid etching action and the striking contrast produced is to be largely attributed.

The areas constituting the mottled etch-pattern which are darkened are those relatively high in phosphorus, as is plainly indicated in Figs. 7 and 8. The ferrite surrounding the iron-phosphide eutectic and that forming part of the eutectic itself constitute the portions which are most strongly colored and not the less contaminated ferrite which solidified first from the melt. This appears to be in direct opposition to the macroscopic results obtained when a sample of wrought iron is etched, in which case copper is precipitated in relatively large amounts upon the layers low in phosphorus. The precipitation of copper upon the streaks of the comparatively pure material is to be attributed to the electrolytic potential of such material with reference to that of copper. The reverse etching of the metal of high phosphorus content cannot be so easily explained upon this basis. The action of the hydrochloric acid alone in the reagent as used will develop the etch-pattern, although much more slowly than when the copper chloride is present. A trial of the two solutions showed

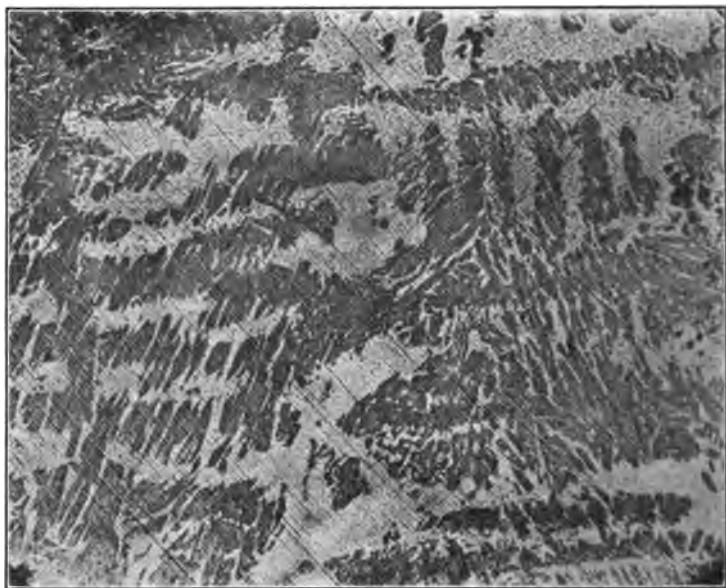


FIG. 7.—ALLOY CONTAINING 0.37 PER CENT. PHOSPHORUS, IN CAST CONDITION ETCHING, ALCOHOLIC COPPER CHLORIDE ACIDULATED WITH HYDROCHLORIC ACID  $\times 25$ .

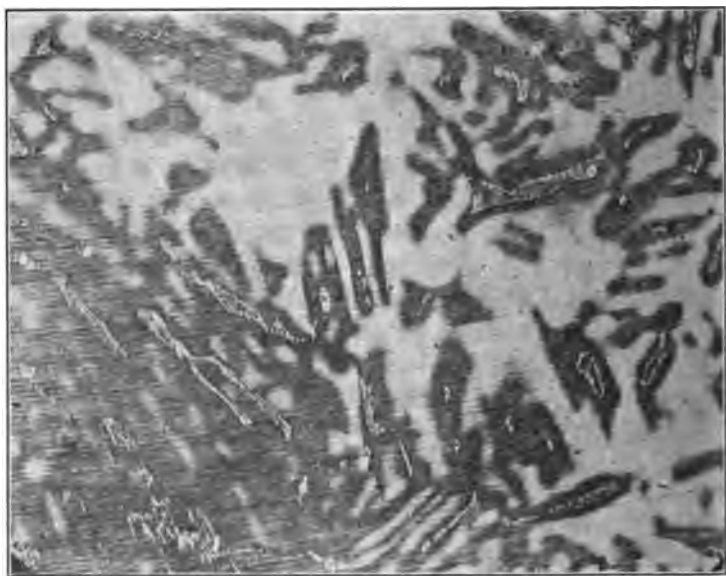


FIG. 8.—SAME MATERIAL AS FIG. 7. SAME ETCHING. THE DARK AREAS ARE PORTIONS OF THE FERRITE HIGH IN PHOSPHORUS, SHOWS SOME OF THE EUTECTIC AS ISLANDS IN THE MIDST OF THE DARK AREAS.  $\times 100$ .

that by a 12-min. immersion in 1 per cent. alcoholic hydrochloric acid a sample of the iron-phosphorus alloy was not etched nearly as satisfactorily as by a 2-min. immersion in the same solution containing copper chloride. The suggestion is offered that metallic copper as such is not deposited but that there is formed a closely adhering thin film of copper phosphide on the phosphorus-rich portions and so the result which acid etching alone will produce if enough time is allowed is hastened and intensified. The roughening of the surface by a prolonged acid etching is thus avoided by the use of the copper-chloride solution.

The etching reagent described by Rosenhain and Haughton<sup>9</sup> was tried upon the iron-phosphorus alloys, but with unsatisfactory results. It should be noted that if the specimens are mounted in some kind of a matrix for polishing, as is necessary for wires, sections of thin pipe, etc., a matrix of soft alloy or anything of metal, must be avoided. The potential difference existing between this metal and the embedded specimen is often so much greater than that between the various portions of the sample under examination as to prevent entirely the differential etching desired.

#### *Behavior upon Heating*

The persistence of the unusual intracrystalline structural features upon heating is remarkable. Fig. 11 shows the appearance of a specimen of the wrought-iron eye-bar previously described after heating for 3 hr. at approximately 600° C. (585°–625°) and then allowed to cool in the furnace. No appreciable changes have resulted by this treatment. A second sample (Fig. 12), heated for about 1½ hr. at approximately 725° C. (718°–735°) and furnace-cooled, still shows faint traces remaining of the former condition. Fig. 13 shows the structure resulting from heating a wrought iron high in phosphorus for 1½ hr. at approximately 700° C.; the specimen being allowed to cool in the furnace. The specimen had the structure shown in Figs. 9 and 10, before heating. The eutectic disappeared by solution of the compound, Fe<sub>3</sub>P, in the ferrite but the non-homogeneity of structure as shown by the dendritic pattern still persists. This illustrates well the remarkably slow rate of diffusion within the ferrite matrix by which equilibrium is finally attained. The presence of the compound Fe<sub>3</sub>P which had separated out from the matrix most probably accounts for the much slower rate of diffusion in this latter case than was observed in the case of the first wrought-iron samples that were annealed.

The slow rate of diffusion of phosphorus in ferrite has often been remarked and it is undoubtedly due to this cause that the mottled structure persists and is not wiped out during the manufacture of the wrought iron, i.e., during the heating, rolling, and forging necessary before the wrought iron reaches the finished condition.

<sup>9</sup> W. Rosenhain and J. L. Haughton: *Journal of the Iron & Steel Institute* (1914, No. 1), 515.

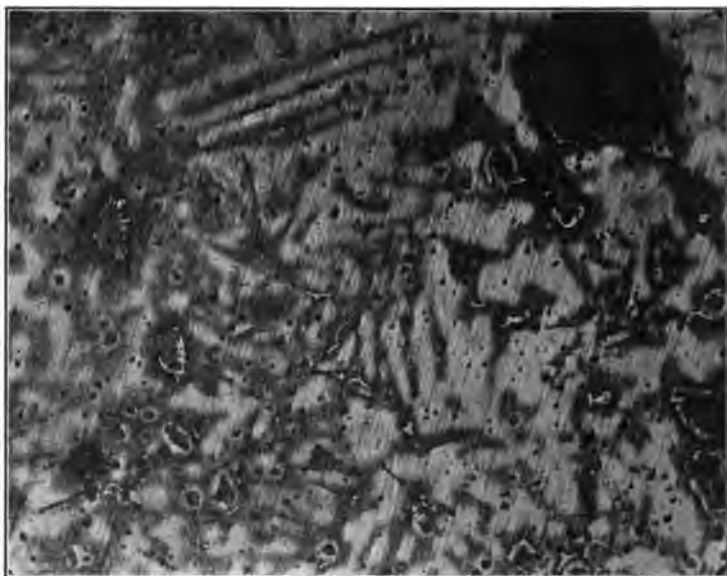


FIG. 9.—HIGH PHOSPHORUS WROUGHT IRON. IN PORTIONS OF THE MATERIAL, ISLANDS OF IRON PHOSPHIDE ARE FOUND, SUCH ISLANDS OCCUR IN THE CENTER OF THE AREAS WHICH BECOME BROWN UPON ETCHING.  $\times 100$ .



FIG. 10.—SAME MATERIAL AS FIG. 9. ETCHING IN BOTH CASES, ALCOHOLIC COPPER CHLORIDE CONTAINING HYDROCHLORIC ACID.  $\times 500$ .

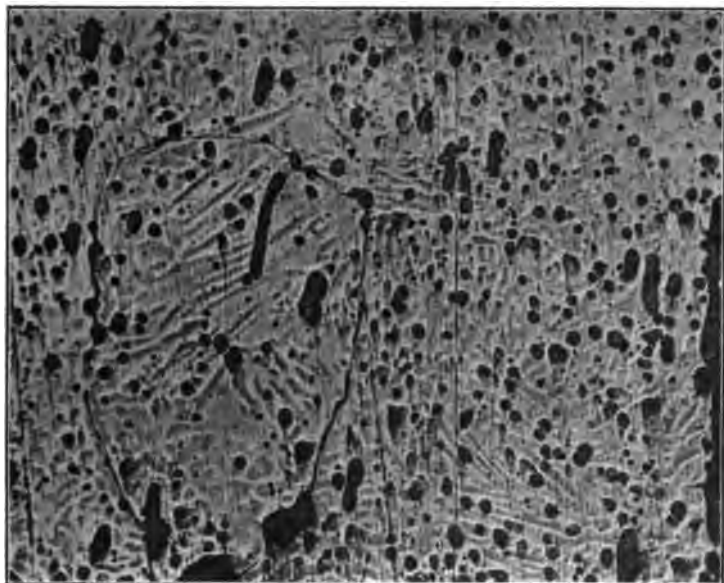


FIG. 11.—EFFECT OF ANNEALING. MATERIAL OF FIG. 3 HEATED FOR 3 HR. AT 600° C. ETCHING, 10 PER CENT. NITRIC ACID.  $\times 100$ .



FIG. 12.—SAME MATERIAL AS FIG. 11, HEATED FOR  $1\frac{1}{2}$  HR. AT 725° C. ETCHING, 10 PER CENT. NITRIC ACID.  $\times 100$ .

## SUGGESTED SIGNIFICANCE OF UNUSUAL FEATURES OF STRUCTURE

The significance of the unusual features of microstructure described and their possible relation to the service behavior of such material may be suggested. The occurrence of material of this type in two of the samples examined (*W1* and *W37*) both of which failed in service and which have the appearance of having failed under the action of alternations or repetitions of stress suggests a possible relation between this type of structure and the failure of such material.

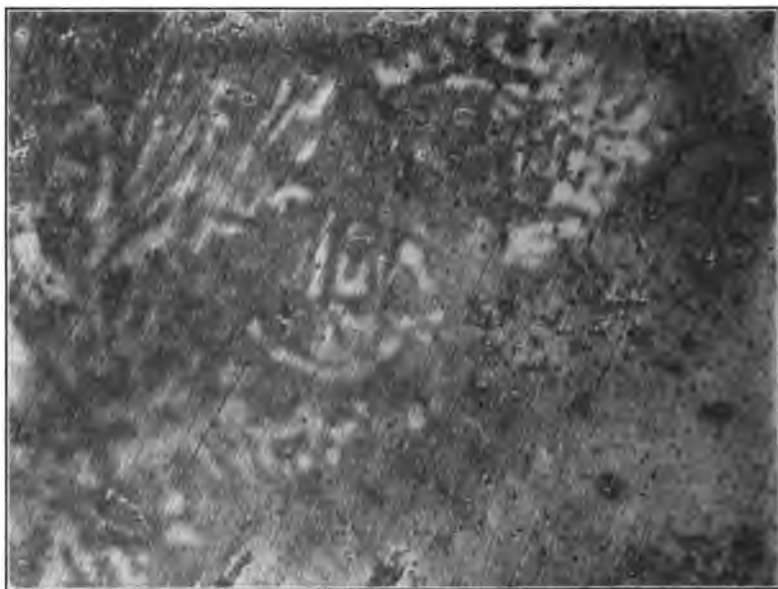


FIG. 13.—HIGH PHOSPHORUS WROUGHT IRON AFTER ANNEALING. MATERIAL OF FIG. 9 AFTER HEATING  $1\frac{1}{2}$  HR. AT  $700^{\circ}\text{C}$ . THE NON-HOMOGENEITY OF STRUCTURE STILL PERSISTS THOUGH THE ISLANDS OF IRON PHOSPHIDE HAVE BEEN DISSOLVED. ETCHING, ALCOHOLIC COPPER CHLORIDE ACIDULATED WITH HYDROCHLORIC ACID.  $\times 100$ .

The method by which "fatigue" breaks occur in metals by the action of repeated stresses has been clearly and conclusively set forth by Ewing and Humphrey,<sup>10</sup> Rosenhain,<sup>11</sup> and others. The minute back-and-forth slip along certain planes occurring within the crystals if repeated a sufficient number of times becomes a permanent displacement, thus initiating an incipient fracture within the crystal. The combined effect of this action within a number of neighboring crystals at some portion of the specimen will be sufficient to cause a real fracture to start at that point if the application of the stresses is continued.

<sup>10</sup> J. A. Ewing and J. C. W. Humphrey: *Philosophical Transactions, Royal Society of London* (1902), 200A, 241.

<sup>11</sup> W. Rosenhain: *Introduction to the study of Physical Metallurgy*, Chapter VIII, New York, 1915.]

The brittle character of ferrite containing considerable phosphorus is well known. Crystals which show the heterogeneity caused by high- and low-phosphorus bands in juxtaposition should be much more easily "fatigued" by repeated stresses and show a permanent slip much more quickly than crystals which are more uniform throughout in their structure. In particular, this should be true if the bands are transverse to the direction of the stresses acting. The observations upon the fracture of the broken eye-bar appear to confirm this. Though the evidence shown by the failed member is only "circumstantial," as is always the case in the examination of "metal failures," still so closely does it corre-

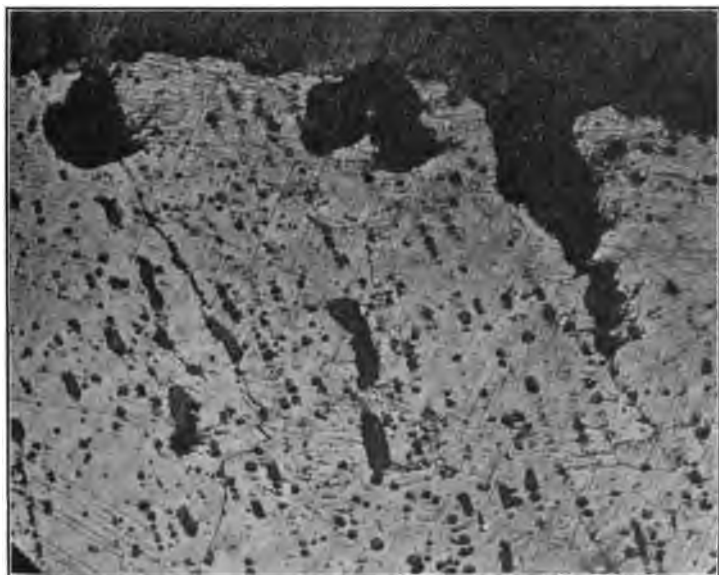


FIG. 14.—RELATION OF STRUCTURAL FEATURES TO THE SERVICE BEHAVIOR OF THE MATERIAL. LONGITUDINAL SECTION OF WROUGHT IRON WHICH FAILED IN SERVICE. THE EDGE OF THE FRACTURE IS PARALLEL TO THE BANDS WHICH CONSTITUTE THE ETCH PATTERN HERE. ETCHING, 10 PER CENT. NITRIC ACID.  $\times 100$ .

spond to and agree with the results of the work of Ewing and Humphrey that there can be but little doubt as to the nature of the fracture.

The fact that the face of the fracture followed and its course apparently was determined by the bands within the ferrite crystals has already been referred to. Fig. 14 shows that the break occurred parallel to the bands of high-phosphorus ferrite which give rise to the peculiar etch-pattern. Examination of the metal immediately back of the face of the fracture reveals further evidence. Fig. 15 shows the appearance of some of the deepest of the etch bands in the crystals close to the break. These, undoubtedly, represent a stage in which minute transverse cracks, i.e., transverse to the direction of the stresses acting, have opened up

within the body of the crystal after a sufficient number of repetitions or reversals of stress. The intracrystalline markings obtained by Ewing and Humphrey by subjecting wrought-iron specimens to repeated alternations of stress in the Wöhler test appear in all respects identical with those shown in Fig. 15.

H. H. Campbell<sup>12</sup> quotes the statement that phosphorus up to 0.20 per cent. is not injurious in wrought iron. This should be taken, however, as referring to an average content of this amount, uniformly diffused throughout the metal. The microsegregation of this element may result in the amount in certain streaks being considerably in excess of

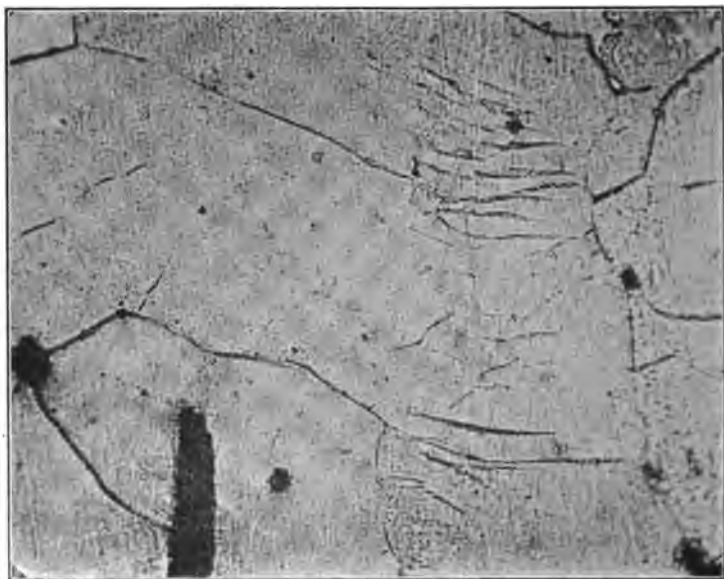


FIG. 15.—APPEARANCE OF VERY PERSISTENT ETCH LINES JUST BACK OF THE FRACTURE IN MATERIAL OF FIG. 14. THE SPECIMEN HAS BEEN TREATED 3 HR. AT 600° C. ETCHING, 10 PER CENT. NITRIC ACID.  $\times 400$ .

this quantity, while within the individual crystals of such streaks the non-homogeneity with respect to phosphorus is still further accentuated. In the material of Fig. 8 the ferrite immediately surrounding the eutectic has a phosphorus content of approximately 1.7 per cent., or nearly five times the average percentage of phosphorus of the sample. It appears very probable then that in such non-homogeneous crystals as are shown in Figs. 1, 3, 4, 5, 9 and 10, the phosphorus content of the bands may be as much as four or five times the average of the whole. In exceptional cases, portions of a sample high in phosphorus may show traces of iron phosphide that has crystallized out directly from the melt (Figs. 9

<sup>12</sup> H. H. Campbell: *The Manufacture and Properties of Steel*, 91, New York, 1896.



and 10). The metal at such points has a phosphorus content not far below 1.7 per cent., as indicated in Fig. 6.

The straight lines, resembling polishing scratches, shown by some of the crystals near the fracture (Fig. 1), are the well-known Neumann lines. Such markings are usually considered to be "mechanical" twin crystals; *i.e.*, the twinned position of the metal within each of the narrow zones having been brought about by mechanical causes alone.<sup>13</sup> They, in all probability, are the result of shock which finally caused the fracture of the bar. A comparison of these lines with the long narrow bands which often constitute the mottled etch-pattern shows immediately that such etch bands cannot be attributed to twinning or a similar cause.

## VI. SUMMARY

1. Wrought irons high in phosphorus sometimes show a peculiar mottled or banded intracrystalline pattern, which by comparison with alloys of pure iron and phosphorus is shown to be due to a non-homogeneous diffusion of the phosphorus.

2. The examination of a wrought-iron member exhibiting such an unusual structure showed that the break which occurred in service bore a definite relation to such banded markings and apparently was largely determined by them.

3. By the non-homogeneity in the distribution of phosphorus throughout the ferrite crystals, the ill effects of phosphorus may be much enhanced.

4. The examination of a series of wrought irons showed that such features are not to be regarded as common. Many poor grades of iron may be unsuitable for other reasons.

5. A second type of markings is described and illustrated—the well-known Neumann lines—and shown to bear no apparent relation to the other variations in structure described.

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<sup>13</sup> H. M. Howe and A. G. Levy: *Trans.* (1915), **51**, 891.

TABLE 1.—*Materials Examined*

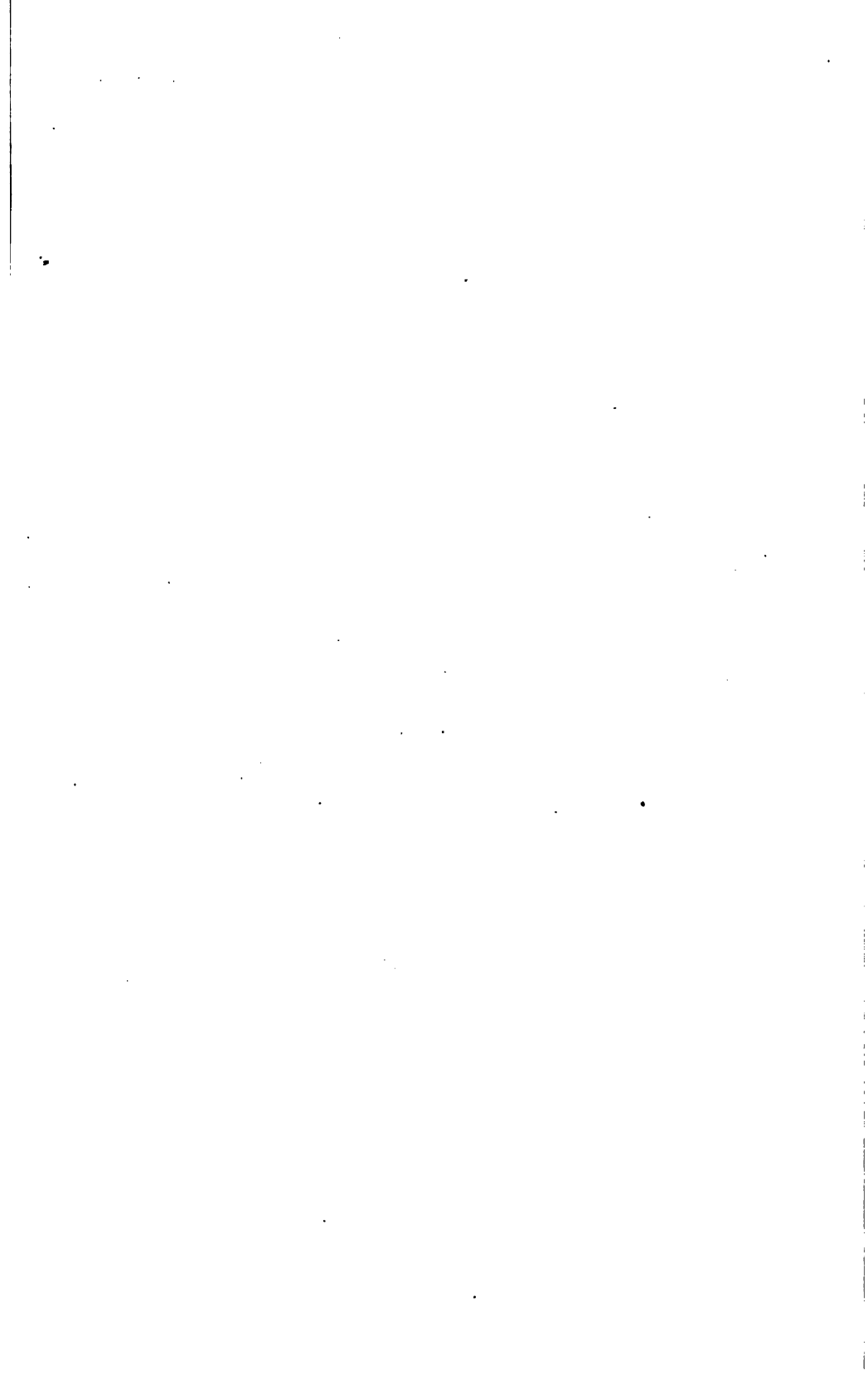
| Sample No. | Type of Material                               | Macroscopic Appearance after Cuprio-Chloride Etching                                       | Microstructure (a)   | Percentage of Phosphorus |
|------------|--|--|--|--------------------------|
| W 1        | Wrought-iron eye-bar, which failed in service. | No copper deposited, surface becomes brown.  | Very prominent mottled streaks.  | 0.356                    |
| W 2        | Wrought-iron pipe.                             | Copper deposited uniformly over surface.   | No mottled areas seen. The metal is to be classed as steel rather than wrought iron. |                          |
| W 3        | Wrought-iron pipe.                             | Copper deposited uniformly over surface.   | No mottled areas seen. The metal is to be classed as steel rather than wrought iron. | 0.130(b)<br>0.200        |
| W 4        | Wrought-iron pipe.                             | Isolated streaks were copper-coated, remainder brown.                                      | Very decided mottled structure in the brown streaks.                                 |                          |
| W 5        | Wrought-iron pipe.                             | Outer layers slightly copper-coated, central ones slightly brown.                          | Mottled streaks very prominent in central portion.                                   | 0.264(b)<br>0.271        |
| W 6        | Wrought-iron pipe.                             | No copper deposited, surface rather brown.   | Mottled streaks very prominent in central portion.                                   |                          |
| W 7        | Wrought-iron pipe.                             | No copper deposited, prominent brown streaks.  | Mottled appearance across entire face of the section.                                | 0.059                    |
| W 8        | Wrought-iron pipe.                             | No copper deposited, brown coloration of surface.  | Mottled appearance across entire face of the section.                                |                          |
| W 9        | Wrought-iron "welding" wire.                   | Copper deposited over all the section.   | No mottled areas seen.   | 0.146(b)<br>0.156        |
| W10        | Wrought-iron (?) bar.                          | Copper deposited over all the section.   | No mottled areas seen. The material is really a low-carbon steel.                    |                          |
| W11        | Wrought-iron pipe.                             | No copper deposited, central streaks very brown.   | Mottled appearance of central streaks very conspicuous.                              | 0.123                    |
| W12        | Wrought-iron pipe.                             | Copper precipitated on outer layers, brown streaks in central part.                        | No mottled areas found.  |                          |
| W13        | Wrought-iron welding wire.                     | Uniform precipitation of copper over the section.  | No mottled areas found.  | 0.160                    |
| W14        | Wrought-iron welding wire.                     | Inconspicuous streaks in central portion, otherwise quite uniform precipitation of copper. | No mottled areas found.  |                          |
| W15        | Wrought-iron welding wire.                     | Inconspicuous streaks in central portion, otherwise quite uniform precipitation of copper. | No mottled areas found.  | 0.160                    |
| W16        | Wrought-iron pipe.                             | No copper precipitated, dark central streaks.  | Streaks of mottled structure abundant across the section.                            |                          |
| W17        | Wrought-iron stay bolt.                        | Some light areas showing method of fagotting, precipitation of copper quite uniform.       | Very slight trace only of mottled areas.   | 0.160                    |
| W18        | Wrought-iron stay bolt.                        | Some light areas showing method of fagotting, precipitation of copper quite uniform.       | Very slight trace only of mottled areas.   |                          |
| W19        | Wrought-iron welding wire.                     | Copper uniformly precipitated over surface.  | No trace of mottled structure found.   |                          |

TABLE 1.—(Continued)

| Sample No. | Type of Material  | Macroscopic Appearance after Cupric-Chloride Etching                                       | Microstructure(a)  | Percentage of Phosphorus |
|------------|---|--|--|--------------------------|
| W20        | Wrought-iron boiler tube which corroded badly in service.   | Copper uniformly precipitated over surface.  | No trace of mottled structure found.   | 0.033                    |
| W21        | Wrought-iron boiler tube which corroded badly in service.   | Copper uniformly precipitated over surface.  | No trace of mottled structure found.   |                          |
| W22        | Low-carbon steel pipe.  | Central streaks free from copper.  | Mottled structure very evident in central layers.  |                          |
| W23        | Wrought-iron "goose-neck."  | Copper deposited quite uniformly over surface.   | No mottled areas found, the metal resembles low-carbon steel rather than wrought iron.   | 0.044                    |
| W24        | .....   | About 0.7 copper-covered.  | .....  | 0.055                    |
| W25        | .....   | About 0.1 reddened by copper, other portions show prominent brown streaks.                 | .....  | 0.136                    |
| W26        | .....   | About 0.9 copper-covered.  | .....  | 0.079                    |
| W27        | .....   | About 0.7 copper-covered, with outer portion clear.  | .....  | 0.059                    |
| W28        | .....   | About 0.9 copper-covered.  | .....  | 0.073                    |
| W29        | .....   | Isolated copper-covered streaks aggregating 0.3 total area, other portions clear or brown. | .....  | 0.112                    |
| W30        | Wrought-iron round stock for forging purposes (chains, etc.) All but W28 and W34 were reported to give unsatisfactory results in the preliminary tests of the material. | Surface clear with brown streaks with one small central copper-covered streak.             | No mottled areas were found even in those showing but little precipitation of copper, except Nos. 29, 30, and 36, which show a slight trace. | 0.099                    |
| W31        | .....   | Only a few narrow streaks are copper-colored, about 0.1 of area.                           |  | 0.115                    |
| W32        | .....   | 0.1 copper-covered, remainder clear with brown streaks.                                    |  | 0.113                    |
| W33        | .....   | One fine streak only, copper-coated, prominent dark streaks in remainder.                  |  | 0.125                    |
| W34        | .....   | Less than 0.5 is copper-covered, brown streaks in the clear portion.                       |  | 0.115                    |
| W35        | .....   | 0.8 to 0.9 of area is copper-covered.  |  | 0.066                    |
| W36        | .....   | 0.6 area copper-coated brown streaks in remainder.   |  | 0.089                    |
| W37        | Link of wrought-iron chain which failed in service.   | No copper deposited, surface was colored brown.  | The mottled etch pattern is very prominent.  |                          |

(a) Under "microstructure" the presence of streaks showing the mottled structure as shown in Figs. 3, 4 above is noted, and not a complete account of the microstructure.

(b) 1 indicates the entire cross-section: 2 the central layer, approximately one-third, the cross-section.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Supposed Reversal of Inheritance of Ferrite Grain Size from that of Austenite

BY HENRY M. HOWE,\* A.B., A.M., S.B., LL.D., SC.D., BEDFORD HILLS, N. Y.

(St. Louis Meeting, October, 1917)

THE data which are collected in Table 1 show that the ferrite of low-carbon steel and of electrolytic iron, like the network of hypo- and hyper-eutectoid carbon steel, inherits, either absolutely or relatively, the grain size of the mother austenite from which it is born in cooling through the transformation range. This evidence is so abundant and concordant as to cast doubt on Prof. Jeffries'<sup>1</sup> hypothesis of what might be called reversed inheritance, by which he explains Mr. Ruder's observation<sup>2</sup> that the grains of a "slug of pressed pure electrolytic iron powder" were coarser, perhaps  $12,000,000\mu^2$ , after a 3-hr. exposure to  $1000^\circ\text{C}$ . than after a 3-hr. exposure to  $1250\text{--}1300^\circ$ , when they were about  $35,000\mu^2$ . His explanation implies a natural tendency of coarser austenite grains to yield finer ferrite ones than spring from finer austenite grains, so that the inheritance is reversed.

What seems to me a simpler explanation is that the enormous grains which followed the  $1000^\circ$  heating were not inherited from the mother austenite, but were formed by the process of natal coarsening, by means of which Stead and Carpenter developed grains of this same order of magnitude in this same kind of material, on heating to  $950^\circ$  or to  $1000^\circ$ , the temperature used by Mr. Ruder, provided the exposure was not unduly prolonged, and it does not seem to have been in Mr. Ruder's case. That their coarsening was natal and not through inheritance is shown first by the fact that their grain coarseness was out of all proportion to their relatively low temperatures and short periods, and hence was very far in excess of the grain size which the mother austenite must have had, and second and more cogently by the fact that the shape of these coarsened grains was evidently determined by the temperature distribution which existed during and immediately after the  $A_r$  transformation, that during the passage from the gamma to the non-gamma state.

<sup>1</sup> Bulletin No. 124 (April, 1917), 495 and No. 126 (June, 1917), 991.

<sup>2</sup> Bulletin No. 124 (April, 1917), 492-3.

In discussing this subject, we must remember that coarse ferrite grains, whether those formed in cooling from the molten or those induced by Stead and Carpenter's process of natal coarsening, break up on transforming into austenite at  $A_{c3}$  on any later heating, the coarsening of ferrite not being inherited by the austenite. Further, that the size of the new ferrite grains formed in case natal coarsening does not occur, depends on the grain size not of the initial ferrite but of the mother austenite, and hence increases with the length and height of heating beyond  $A_{c3}$ . Hence, in Mr. Ruder's second or  $1300^{\circ}$  heating, the coarse initial grains of ferrite formed natively in the  $A_{r3}$  transformation after his prior  $1000^{\circ}$  heating, and representing not that heating but that transformation, broke up on rising past  $A_{c3}$  in his next heating, which was to  $1300^{\circ}$ . The ferrite grains which would next form on cooling past  $A_{r3}$  would inherit the austenite grain size formed in this  $1300^{\circ}$  heating, and would have no tendency to

TABLE 1.—Selected Cases of Grain Growth

| No.                            | Date | Investigator     | Carbon Content, Per Cent. | Behavior of Specimen in Prior or Later Treatment | Behavior of Specimen in Present Treatment |                    |        |                               |  |  |
|--------------------------------|------|------------------|---------------------------|--|---|--------------------|--------|-------------------------------|--|--|
|                                |      |                  |                           |  | Treatment                                 |                    |        | Average Grain Size in $\mu^2$ | Summary of Coarsening  |  |
|                                |      |                  |                           |  | Exposed to $^{\circ}\text{C}.$            | Length of Exposure |        |                               |  |  |
|                                | Hr.  | Min.             |                           |  |   |                    |        |                               |  |  |
| GROUP I. HEREDITARY COARSENING |      |                  |                           |  |   |                    |        |                               |  |  |
| A. Low-Carbon Steel            |      |                  |                           |  |   |                    |        |                               |  |  |
| 1                              | 1900 | Fay and Badlam   | 0.07                      |  | 1,143                                     | ..                 | ...    | 600                           | Slight to 1,125° then greater to 1,247°                                |  |
| 2                              | 1900 | Fay and Badlam   | .....                     | .....  | 1,247                                     | ..                 | ...    | 20,000                        |  |  |
| 3                              | 1911 | Howe             | 0.027                     |  | 1,200                                     | 1                  | ...    | 6,900                         | Slight at 1,200°, marked at 1,300°.                                    |  |
| 4                              | 1911 | Howe             | .....                     | .....  | 1,300                                     | 1                  | ...    | 37,000                        |  |  |
| 5<br>6<br>7                    | 1916 | Pomp.....        | 0.08                      | .....  | { 1,100<br>1,200<br>1,300                 | 8                  | ...    | 1,480                         | Slight and irregular up to 1,100°, great and nearly regular at 1,300°. |  |
| 8                              |      |                  |                           |  |   | ...                | 16,090 |                               |  |  |
| 8                              |      |                  |                           |  |   | ...                | 17,350 |                               |  |  |
| B. Electrolytic Iron           |      |                  |                           |  |   |                    |        |                               |  |  |
| 8                              | 1909 | Terry.....       | .....                     | .....  | 1,300                                     | ?                  | ...    | 20,000 $\pm$                  | Slight coarsening at 1,000°, far more at 1,200° and 1,300°.            |  |
| 9                              | 1911 | Howe.....        | 0.023                     | .....  | 1,300                                     | ..                 | ...    | 9,500                         |  |  |
| 10                             | 1911 | Howe.....        | 0.029                     | .....  | 1,300                                     | ..                 | ...    | 9,800                         |  |  |
| 11<br>12<br>13                 | 1914 | Storey Ingot B { | .....                     | .....  | 950                                       | ..                 | 30     | 7,400                         |  |  |
| .....                          |      |                  | .....                     | 1,050  | ..  | 15                 | 23,000 |                               |  |  |
| .....                          |      |                  | .....                     | 1,150  | 3   | ...                | 22,900 |                               |  |  |
| 14                             | 1917 | Ruder.....       | .....                     | Previously Coarsened at 1,000°, see No. 29.      | 1,250-1,300                               | 3                  | ...    | 35,000                        |  |  |

\*Emeritus Professor of Metallurgy, Columbia University.

TABLE 1.—*Selected Cases of Grain Growth (Continued)*

| No.   | Date | Investigator        | Carbon Content Per Cent.                            | Behavior of Specimen in Prior or Later Treatment | Behavior of Specimen in Present Treatment |                    |    |  |                        |  |
|---|------|---------------------|---|--|---|--------------------|----|--|------------------------|--|
|   |      |                     |   |  | Treatment                                 |                    |    | Average Grain Size in $\mu^2$                        | Summary of Coarsening  |  |
|   |      |                     |   |  | Exposed to °C.                            | Length of Exposure |    |  |                        |  |
|   |      | Hr.                 | Min.  |  |   |                    |    |  |                        |  |
| GROUP II. THE NATAL COARSENING OF ELECTROLYTIC FERRITE        |      |                     |   |  |   |                    |    |  |                        |  |
| 15  | 1913 | Stead and Carpenter | 0.008   | .....  | 950                                       | ....               | 30 | Cooled to 885 then quenched.                         | Fine                   | Fine if quenched above $A_{r_2}$ , coarse if quenched below. |
| 16  | 1913 |                     | 0.008   | .....  | 950                                       | ....               | 30 | Cooled to 880 then quenched.                         | Coarse                 |  |
| 17  | 1913 |                     | 0.008   | .....  | 1,000                                     | .....              |    | Quenched at 1000.                                    | 4,400,000 Fine         |  |
| 18  | 1913 |                     | 0.008   | .....  | 1,000                                     | .....              |    | Quenched 2 or 3 sec. after cooling through $A_{r_2}$ | Coarse                 |  |
| 19  | 1913 |                     | 0.008   | .....  | 890                                       | 72                 |    |  | Fine                   |  |
| 20  | 1918 |                     | 0.008   | .....  | 910                                       | 1                  |    |  | 2,700,000 <sup>a</sup> |  |
| 21  | 1913 |                     | 0.008   | .....  | 915                                       | 1                  |    |  | 7,000,000              |  |
| 22  | 1913 |                     | 0.008   | .....  | 930                                       | 1                  |    |  | 11,300,000             |  |
| 23  | 1913 |                     | 0.008   | .....  | 950                                       | 1                  |    |  | 9,750,000              |  |
| 24  | 1913 |                     | 0.008   | .....  | 1,000                                     | .....              |    | Heated 6 times to 1,000.                             | 26,000,000             |  |
| 25  | 1912 |                     | 0.008   | .....  | 915-930                                   | ....               | 1  |  | 4,300,000              |  |
| 26  | 1913 |                     | 0.008   | .....  | 950                                       | 1                  |    |  | Coarse                 |  |
| 27  | 1913 |                     | 0.008   | Coarsened in 1 hr. at 950°, No. 26.              | 950                                       | 21                 |    |  | Fine                   |  |
| 28  | 1913 | Ruder....           | 0.008   | Refined in 21 hr. at 950°, No. 27.               | 950                                       | 1                  |    |  | Coarse                 |  |
| 29  | 1917 |                     | Later refined by heating at 1250-1300°. see No. 14. | 1,000  | 3   |                    |    | 1,500,000 <sup>a</sup>                               |                        |  |
| GROUP III. COARSENING OF FERRITE BY THE COMPETITIVE PRINCIPLE |      |                     |   |  |   |                    |    |  |                        |  |
| 30  | 1917 | Ruder....           | Silicon steel                                       | .....  | 1,300                                     | ....               |    | Fed slowly past 1,300°                               | 9,500,000.-000 nearly  |  |

resume the now effaced enormously greater size given natively during the prior  $A_{r_2}$  transformation. I do not mean to assert that the austenite grain size is wholly unrelated to that of the pre-existing ferrite, but rather that the grain-size relation of parent to offspring is far closer in the austenite-ferrite or  $A_{r_2}$  than in the ferrite-austenite or  $A_{c_2}$  transformation.

It is indeed surprising that Mr. Ruder's electrolytic specimen should undergo this enormous natal coarsening, because its thickness was probably far beyond the limiting thickness, about 0.012 in. (0.3 mm.), which Stead and Carpenter found sufficient to bar natal coarsening. His coarse grains have not only their size but also their ragged outline and irregular shape to show that they are to be classed with the natively coarsened grains of Stead and Carpenter. They are shown thus not only by their size but by their shape and outline to be formed natively and not inherited from the mother austenite.

<sup>a</sup> This is the size of one grain, not the average grain size.

THE UNIVERSITY OF CHICAGO  
CHICAGO, ILLINOIS  
JANUARY 1961

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1961



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## The Coal Industry of Illinois

BY C. M. YOUNG,\* B. S., E. M., URBANA, ILL.

(St. Louis Meeting, October, 1917)

THE following paper has been prepared with the object of placing on record in the *Transactions* some facts concerning the present condition and future prospects of the coal industry of Illinois. In presenting it, the writer wishes to say that a considerable amount of the material contained is taken from the publications and records of the Illinois Coal Mining Investigations, the annual Coal Report of the State Mining Board, and the work of the State Geological Survey.

### *Coal Beds and Mining Methods*

According to the latest maps of the State Geological Survey, there are only 19 of the 102 counties of the State some portion of whose territory is not within the borders of the coal measures. If outlying areas in Calhoun and Whiteside Counties are considered, the number is reduced to 17. This does not mean that all of the 85 counties that are partly or wholly underlain by coal measures will become producers, as there are some portions of the State in which the coal does not occur in beds of such thickness as to warrant the hope that it can be mined. The mining of coal is widespread, however, for in 1916 there was production from 51 of the 102 counties of the State. There seem to be no cases in which operations will be limited by depth, as none of the coal measures lie at depths much greater than those at which it has been proved that coal may be profitably mined in this field. In fact, the deepest shaft reaching a bed of bituminous coal in the United States is located in this State at Assumption, where coal only  $3\frac{1}{2}$  ft. (1.07 m.) thick is worked at a depth of 1004 ft. (306 m.).

The coal beds occur in the form of a basin which is deepest along a line extending through the central part of the State and bearing a little west of north. The northern, western and southern borders of this basin lie within the boundaries of the State, but on the eastern side the Illinois field is continuous with the Indiana field. The glacial drift covers all except the extreme southern end of the State and the visible outcropping

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of the coal is much less frequent than it would be if it were not concealed by this material.

The older beds are found over a somewhat larger area than the more recent beds; for example, coal No. 1 is worked in Rock Island County at the extreme northwestern part of the coal area and as far south as Monmouth in Warren County. Southeast of this district are places where No. 2 is worked, as at Colchester, Macdonough County, and Augusta, Hancock County. Further to the southeast, workings are found in beds 5, 6, and 7.

The areas in which the different beds are most largely worked are shown in Fig. 1, a map showing the division of the State into districts for the purposes of the Illinois Coal Mining Investigations. These investigations are being carried on in coöperation by the State Engineering Experiment Station, the State Geological Survey, and the United States Bureau of Mines. The districts are areas in which certain coals are worked under fairly uniform conditions. The districts and the beds worked in each are as follows:

| District | Bed     |  |
|----------|---------|--|
| 1        | 2       | Longwall                                     |
| 2        | 2       | Room-and-pillar                              |
| 3        | 1 and 2 | Room-and-pillar                              |
| 4        | 5       | Room-and-pillar                              |
| 5        | 5       | Room-and-pillar                              |
| 6        | 6       | East of Duquoin anticline<br>Room-and-pillar |
| 7        | 6       | West of Duquoin anticline<br>Room-and-pillar |
| 8        | 6 and 7 | Room-and-pillar and stripping.               |

Most of the coal of the State is produced from beds 1, 2, 5, 6, and 7. The production from bed No. 1 is small, amounting to only about a half million tons. If the indefinitely correlated coal at Assumption is excepted, the working of the No. 1 coal is confined to the territory in the neighborhood of Rock Island. In this district the average thickness of the coal is 4 ft.

It should be said that the term "No. 1" is somewhat indefinite in its application. Coal No. 2 is quite well marked, and anything occurring below it is called No. 1, though it is not certain that all coals called No. 1 belong to exactly the same horizon.

The production from bed No. 2 during the fiscal year ending June 30, 1916, was 4,305,330 tons. This bed is more widely distributed than any other and is to be found, so far as is now known, over the whole coal area of the State with the exception of a few counties in the east-central part. The coal is now mined in 16 counties, and if the Assumption mine is in No. 2 coal, in 17 counties.

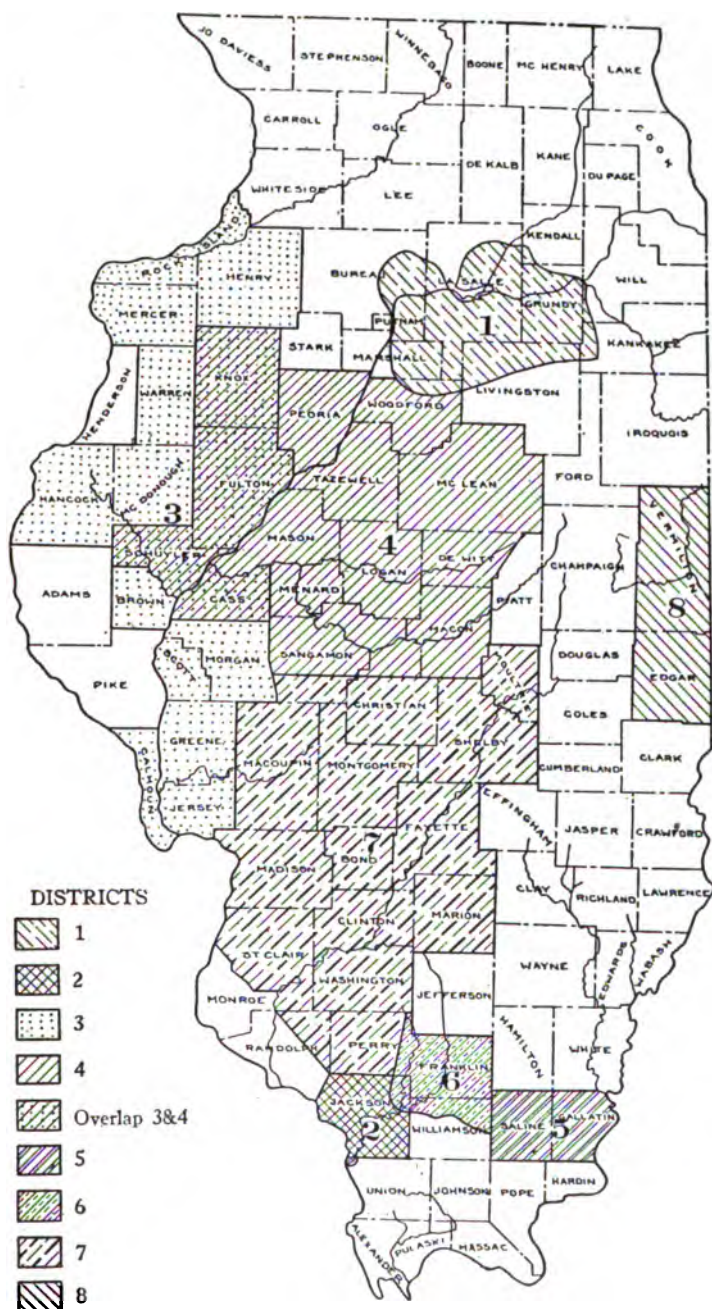


FIG. 1.

The coal is of good quality for coals of this field, but it is not always of workable thickness, and is nowhere very thick. The greatest thickness is probably in Jackson County, where the two benches together have an average thickness of 5 ft. 9 in. (1.75 m.). In the longwall field (District 1) the average thickness is 3 ft. 2 in. (0.96 m.).

The No. 2 coal is most largely worked along the northern edge of the field in Bureau, LaSalle, Grundy, and Will Counties. It is also worked along the western border of the field, in Warren and McDonough Counties at the north, and especially in the neighborhood of Murphysboro, Jackson County, at the south. In this latter district the bed is divided into two benches by a parting varying from  $\frac{1}{8}$  in. to 36 ft. (3.175 mm. to 10.9 mm.) in thickness. The average thickness of the bottom bench is  $3\frac{3}{4}$  ft. (1.15 m.) and that of the top bench 2 ft. (0.6 m.). Where the parting is less than 4 in. (101.6 mm.) thick the two benches of the seam are worked as one, but where it exceeds this thickness only the lower bench is worked. The area over which the two benches are worked together is approaching exhaustion, though there still remains a large area over which one bed alone can be worked. The coal of this district is better in quality than that from most other parts of the State, and finds a ready market at a higher price than that commanded by most of the other coals.

Coal beds are most numerous in the southern part of the State, and some mining on a small scale is done in some of the lower coals which outcrop to the south of the outcrop of No. 6. Coal thus mined is called No. 3. The same number is also popularly associated with the coal mined at Roanoke in Woodford County, and at Pleasant Plains in Sangamon County. However, the terms "No. 3" and "No. 4" do not belong to any definitely known coals, and no mining is done in any coal known as No. 4.

No. 5 is a well-known bed occupying a large territory. It is extensively worked in the north-central part of the State and is again found of good thickness and quality in Saline and Gallatin Counties in the southeast part of the State.

Of the No. 5 coal there was produced in the fiscal year ending June 30, 1916, 12,986,274 tons. As found in the north-central part of the State, the average thickness of this coal is 4 ft. 8 in. (1.42 m.). One of the characteristics of this district is the large number of clay veins crossing the coal bed. There are also small falls, slips, and rolls. There are numerous cases where the coal has been eroded and the space filled with drift. These conditions make it difficult to estimate the total contents of the bed in any area before it has been thoroughly developed.

In Saline and Gallatin Counties the No. 5 coal is of variable thickness, averaging 5 ft. 4 in. (1.63 m.) in Saline County and 4 ft. (1.22 m.) in Gallatin County. The No. 6 coal is also worked to some extent in

Gallatin County, and one shaft is to be sunk from No. 6 to No. 5, thus permitting the working of both beds. There are also in Saline County some operations in what is locally known as Coal No. 7, but this coal is really No. 6. In Gallatin County there are also some operations in the No. 6 coal and one in what is known as No. 9 coal.

In southern Gallatin County there is a district in which several beds of coal are found. These have not yet been correlated, and it is impossible at present to assign definite names to all of them, but such investigation as has been made shows that some of these coals at least are of excellent quality, better in fact than the No. 2 coal as known at present. This district has not yet been extensively developed, partly because the strata are somewhat folded and the conditions are unfamiliar to Illinois operators, and partly because there is no railroad to handle the coal. A railroad is now being built, and it is probable that coal from this district will soon reach the market.

All through the south-central part of the State the No. 6 coal is worked, and this is the bed from which the greatest tonnage is now produced. In Fig. 1, the area in which the No. 6 coal is worked is shown divided into two districts, 6 and 7. This is because the physical conditions are different on the two sides of the Duquoin anticline, which extends approximately north and south across eastern Perry County and into Jackson County. Because the No. 6 coal is of greater thickness and is generally found at greater depth on the east side of the anticline, Franklin County and the northern part of Williamson County have been separately studied. In these counties, and to some extent in Saline County east of them, there has been the most rapid growth of the industry in recent years, and in the same district present activities give promise of the greatest expansion in the near future.

According to E. W. Shaw and T. E. Savage, the thickness of No. 6 coal in District 6 ranges from 7 ft. 6 in. to 14 ft. (2.29 to 4.3 m.), and an average thickness of 9 ft. 9 in. (2.97 m.) was found in 130 borings.<sup>1</sup> There are limited areas, however, in which a thickness of 15 ft. (4.6 m.) or more is found.

A persistent band of impurity, known as the "blue band," is found wherever the No. 6 coal is known. Almost everywhere it is from 18 to 30 in. (0.46 to 0.76 m.) above the floor. No satisfactory explanation has been offered for the persistent occurrence of this bed at almost the same elevation and over a wide area.

Near the top of the No. 6 coal in District 6 is a persistent parting, at which the coal of the main bench is easily separated from the top coal. Neither main bench nor top coal is of uniform thickness. In some places

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<sup>1</sup> Murphysboro-Herrin Folio, Illinois. *U. S. Geological Survey, Folio No. 185, (1912).*

the main bench is only about 6 or 7 ft. (1.8 or 2.2 m.) thick, and in others it reaches perhaps 10 ft. (3 m.). The top coal varies in thickness from about 18 in. (0.46 m.) to about 5 ft. (1.5 m.). The greatest thicknesses of the two benches commonly accompany each other.

There is no uniform difference in quality between the main bench and the top coal. While it is true that neither one is uniformly superior to the other, it has been found that the top coal is occasionally superior to any known coal of the main bench. According to Parr,<sup>2</sup> at certain places where the ash content of the lower bench is 6.41, 6.98, and 9.46 per cent., that of the top coal is only 3.60, 3.05, and 3.12 per cent. respectively. The heat values of the two benches at the same places, "as received," are: top coal, 11,912, 12,786, 12,275; lower coal, 12,000, 12,001, 11,395 B.t.u. These analyses show that there are places in which the top coal is quite noticeably superior to that of the main bench. No attempt has been made to mine and ship these coals separately, but it seems possible that this may be done at some time in places where the top coal is of such high quality as to make it desirable, and especially where the bed is thick. At the present time a considerable part of this top coal is left as a permanent roof.

On the west side of the Duquoin anticline the thickness of the No. 6 coal ranges from  $2\frac{1}{2}$  to 14 ft. (0.76 to 4.3 m.) and averages 7 ft. (2 m.). In this part of the district, also, there is a top coal which is left up where a black shale roof is found and where the coal is 7 ft. thick or more. According to the Coal Report, the production of No. 6 coal in the fiscal year ending June 30, 1916, was 43,011,832 tons.

Near the southern end of the State at approximately latitude  $37^{\circ} 30'$  N., the coal measures are brought to the surface by a general uplift and the coal has been eroded along a line extending a little south and east from the neighborhood of Murphysboro. Some of the coals are found again in Kentucky.

The No. 7 coal is worked to a small extent in the north central part of the State but the principal production is from the neighborhood of Danville, where this coal has an average thickness of 5 ft. (1.5 m.). Special interest attaches to this district because it was here that the first large revolving steam shovel was put in operation and proved so successful that similar and larger shovels were used here and in other districts. The No. 6 coal is worked in the same neighborhood but by ordinary mining methods. The production of No. 7 coal in the fiscal period referred to was 2,649,280 tons.

Outside of the longwall district, most of the operations in the State are conducted with some form of the room-and-pillar system. In most

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<sup>2</sup> S. W. Parr: Chemical Studies of Illinois Coals. Illinois Coal Mining Investigations, *Bulletin* 3 (1915), p. 51.

cases this is either a cross-entry method or a so-called panel method. In the latter, however, there is not often sufficient care in maintaining the isolation of the groups of rooms turned from room entries to warrant the use of the term panel.

One of the advantages to be derived from the application of the panel method is the limitation of squeezes. The mining practice of Illinois has been such that these are of frequent occurrence. It has been and still is the general custom to obtain most of the coal on the advance, with little regard to the extraction of pillar coal. Under these circumstances, an effort is made to obtain as much coal as possible from the rooms and entries, with the result that pillars are left too small to sustain the weight upon them without movement, and squeezes are produced.

In most parts of the State the coal is overlain by one or more beds of rock sufficiently strong to resist breaking in the ordinary course of mining. When a pillar fails, these strong beds transfer the weight of the overlying material to the surrounding pillars, with the result that these are likely to fail. A squeeze thus produced may travel over a considerable distance until it is stopped by some pillar sufficiently strong to bear the weight, or until the strong rock breaks.

One result of such movements is the production of subsidence upon the surface, and, because of the high value of Illinois lands over a considerable part of the State and of the poor surface drainage due to the level topography, subsidences are more noticeable than in most other bituminous districts. For this reason there has been a tendency on the part of coal producers to plan their operations with the idea of leaving in the ground sufficient coal to maintain the surface in its original position. In some cases this appears to have been done successfully at the expense of an average loss in the ground of about 50 per cent. of the coal. In other cases the attempts to prevent squeezes and subsidence have not been successful. It seems very desirable that plans should be adopted in which the subsidence of the rock overlying the coal will be recognized as a necessary accompaniment of mining and in which the damage from movement will be minimized by such planning of the workings as will confine the movement to districts ready to be abandoned. This can probably be done only by making the workings of such dimensions that there will be no danger of subsidence during the advance, and taking out all or nearly all of the coal on the retreat, so that the subsiding top will have no partial support to retard its fall and promote squeezes.

In most cases it is possible by proper tiling and ditching to restore the surface to good condition for agricultural purposes, and the subsidence of the surface should not be considered an obstacle to high extraction except in cases where the coal producers do not own surface rights and are forced to pay for surface subsidence much in excess of a just compensation for the damage done.

In spite of the long line of the boundary of the coal district, there is little stripping except in the Danville district. In most cases this is due to the fact that the surface is covered with glacial drift so that the coal is not exposed. Some stripping is done in the southern part of the State, and it is probable that more will be done, as there is a considerable stretch of territory where the No. 6 coal approaches close enough to the surface to be profitably extracted by this method.

The mining practices in the different districts are for the most part founded upon the cheap production of coal whose value in the ground is low. In the symposium on Studies of Illinois Coals<sup>2</sup> there is given a discussion by George S. Rice of the relation between the low price paid for coal in the ground and the mining methods employed in extracting it. Attention is there drawn to the fact that coal rights are sold for very low prices and that for this reason there has been little incentive, so far as the value of coal in the ground is concerned, to extract the highest possible percentage of the coal. On the other hand, the desire of the producers has been to place coal on the market at so low a price that it could compete with other coals.

Since the publication of this symposium there has been some increase in the values of coal rights, but even yet the value of coal in the ground has not reached such a point as to make high extraction a necessary consideration in the choice of mining methods. Therefore, it is found that over a considerable part of the State wasteful methods are employed and it is safe to say that an average of only about 50 per cent. of the coal in the ground is taken out, and that the remainder is left in such form that it cannot later be extracted at a profit.

Shortly before the beginning of the present abnormal conditions in the industry due to the war, there was evidence of a desire on the part of several coal producers to increase the percentage of coal won, and it is probable that various plans for the increase of the percentage of extraction which have been temporarily abandoned will be taken up again when more nearly normal conditions exist.

An exception to the statement made concerning low extraction is found in the longwall district, whose location is shown in Fig. 1. Formerly beds 2 and 5, known locally respectively as the third and second veins, were worked by room-and-pillar methods, but after a time the longwall system was introduced and since that time the No. 2 bed has been worked by this system while nearly all the operations in the No. 5 coal have been abandoned. It is to be supposed, however, that the No. 5 coal will be worked later.

The extensive development of southern Illinois coals and the establishment of freight rates that permit these more cheaply mined coals to enter

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<sup>2</sup> *Trans.* (1909), 40, 3.



the markets formerly supplied by coal from the northern part of the State, greatly interfered with the prosperity of the longwall district, and the termination of some of the operations was considered a possibility. A recent application of new methods has greatly improved the prospects of the coal industry in this district.

The only remedy for the depressing conditions seemed to be in a decrease of the cost of production, which apparently could be brought about by the introduction of machine mining and electric haulage. According to C. C. Swift,<sup>4</sup> the first attempt at machine mining was made some 20 years ago or more. Not unnaturally this early attempt was not successful. The first experimental work in recent years was commenced in September, 1914, and the machine tried was adopted. Other experiments resulted in improvements in the machines used, and in June, 1917, 11 machines were in operation and some 40 more had been ordered.

Whether the introduction of machines would have been so rapid if the unprecedented demand of the last few months had not occurred is open to question, but the movement had already been started and the use of machines had been decided upon in some cases before there were any indications of the present large demand upon the resources of the district. In fact, electric haulage and machine cutting had been adopted not as a means for meeting an increased demand, but as a measure of economy which would permit the continued operation of the mines under adverse conditions.

### *Growth and Condition of the Industry*

An inspection of the present condition of the coal-mining industry of Illinois and of its past history shows that this industry has not yet entirely emerged from the formative period. It is not unified; forces are constantly pulling in different directions, so that there is lack of harmony and no possibility of wide planning for the most economical conduct of operations along lines leading to full utilization of the coal deposits. Among the operators the desirability of harmonious relations is beginning to be felt, but there is as yet little practical coöperation. The competition between individual operators working in the same parts of the district has been partially eliminated in some cases, though not in all; but there is by no means such a harmonious relation of different districts over the State as would make possible the economical distribution of output and the elimination of a part of the waste due to over-capacity and destructive competition.

It is probable that coöperation among producers will become thoroughly established only after some wide control of the industry has been

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<sup>4</sup> C. C. Swift, General Manager, LaSalle County Carbon Coal Co., "personal communication.

assumed by the State or by the Nation. Such desire for coöperation as is felt is hampered in transformation into effective results by laws intended to prevent improper combinations of interests controlling industries. While these laws were undoubtedly necessary in view of the conduct of some combinations of capital in the past, there can be no question that they operate to prevent such coöperative control of industry in some cases as would work to the ultimate benefit of all parties concerned. At present the public does not trust any combination that seeks to do away with competition, and probably will not until there is such machinery for the just and adequate control of combinations as will certainly and permanently remove all danger of abuse.

It is true that the companies owning the larger resources are in position to plan for long periods of time, but this fact has not yet led to such methods of mining as seem desirable. The endeavor still is to produce coal at what is thought to be the lowest cost, with little if any regard to the waste of immense quantities of coal in the ground. When we realize that the production in the fiscal year ending June 30, 1916, was 63,673,530 tons, and that practically the same amount of coal was left in the ground in an unrecoverable form, we see that the industry is not conservatively conducted.

This is not wholly a reason for criticism of coal operators, for the industry has developed along the line which made its development possible under existing commercial conditions; that is, by the production of coal at so low a cost that it can be sold in competition with other coals.

When there is unrestricted competition in an industry which is occasionally if not always profitable, there is likely to be over-development, as there is in Illinois at present. With over-development there cannot be the most economical working, but an effort is made to produce coal as cheaply as possible without regard to waste of coal in the ground. Some operators are endeavoring to remedy the present conditions and there is no doubt that improved methods of mining will be adopted in the future.

In view of the present demand for American coal and the probable future demand, it is interesting to observe the changes that are taking place and to look into the possibility of maintaining the present rate of output or of increasing it. In the fiscal year ending June 30, 1916, the production of Illinois coal was greater than that of any preceding year but the record can probably be easily exceeded if there is demand for the coal and if the output of the mines can be transported.

A change in the nature of the industry has been going on for some years with regard to the output of individual mines, and this is naturally accompanied by changes in the number of mines. To go back to 1906, at which time the number was greatest, we find that there were 1018 coal mines in the State. By 1916 this number had decreased to 803,

yet the number of men employed in the industry had increased from 62,283 to 75,919, and the amount of coal produced had shown an even greater increase, rising from 38,317,581 tons in 1906 to 63,673,530 tons in the fiscal year ending in 1916.

One of the changes in the industry shown by these figures is in the rate of production per employee. - In 1906 this was 615 tons per man, and in 1916 it was 838 tons per man. It may be noted at the same time that the wages paid per ton have increased in the same period. This change in rate of production per man is probably accounted for principally by two things: first, the increased use of explosives; and second, the increased use of mining machines.

There is apparent a change in the character of the operations with regard to the output from individual mines similar to that which has taken place in some of the older districts; that is, the gradual increase in the average output is accompanied by a decrease in the number of small producers. In other words, there has been an increase of large producers. To make comparisons between the years that have previously been used, the average production in the fiscal year ending June 30, 1906, was 37,640 tons, while in 1916 it was 79,294 tons.

This tendency toward larger production from individual mines is further indicated by the decrease in the number of shipping mines from 419 in the fiscal year ending in 1906 to 284 in 1916. Meanwhile the production from such mines had increased from 37,133,811 tons in 1906 to 62,283,236 tons in 1916, the average output of shipping mines having increased from 88,624 tons to 219,307 tons.

There has been a slight decrease in the number of local mines (that is, mines which do not ship coal by rail), the number being 599 in 1906 and 519 in 1916. The production from mines of this class has increased slightly, being 1,194,770 tons in 1906 and 1,390,294 tons in 1916. The average output from these mines was 1994 tons in 1906 and 2678 tons in 1916. It is apparent that the proportion of coal produced at local mines is constantly decreasing.

The increase in the importance of large producers is further shown by the fact that the number of mines producing 200,000 tons or more has increased in the period under discussion from 47 to 111, or from 4.62 per cent. of the total number to 13.82 per cent.

The distribution of the increased production from the shipping mines is interesting as an indication of the tendency toward large outputs. In the fiscal year ending in 1906, the mines producing 200,000 tons or more per year had a combined output of 12,565,030 tons, while in 1916 the output of this class had reached 48,375,758. No other class of mines showed an increase for this period as a whole, but all classes producing less than 200,000 tons showed a decrease. In other words, the increased production in the State has come entirely from the larger mines. As the

the total increase in this period was 25,355,949, and the increase in production from the larger mines was 35,810,728 tons, we see that these mines have not only produced all of the increased tonnage, but that they have made up for the decrease in production of 10,454,779 tons from mines of the other classes. This does not mean that individual mines of the smaller classes may not have shown increases in production, but that the mines producing 200,000 tons or more are the only ones showing increased production as a class.

This seems to indicate a struggle for existence in which the largest mines are proving victorious. Probably this is in part due to cheaper production from these large mines, but it is probably also due to the financial condition of the companies operating them. The past few years have been a period of very low profits for producers of Illinois coal in general, and many were unable to survive this period without being placed in the hands of receivers, while a considerable number succumbed entirely. This may account for the decreased production from smaller mines because of the decreased number of such mines. The increased production from the larger mines may be due in part to greater ability to dispose of the coal, for it seems, from an inspection of the tables of the Coal Report, that the operation of these smaller mines continued for a smaller number of days than that of the larger mines.

The concentration of production at larger mines is shown by the number of mines whose output in the fiscal year ending June 30, 1916, was 100,000 tons or more. No mine in the State reached an output of 1,000,000 tons, though Superior No. 3, came within less than 4 days' average output of it.

| Production Over<br>Tons | Number of Mines |
|-------------------------|-----------------|
| 100,000                 | 171             |
| 200,000                 | 111             |
| 300,000                 | 74              |
| 400,000                 | 53              |
| 500,000                 | 37              |
| 600,000                 | 22              |
| 700,000                 | 14              |
| 800,000                 | 7               |
| 900,000                 | 5               |

It may be interesting as illustrating the capacities of some of the Illinois mines to mention some of the daily averages and the largest day's outputs. Considering the fiscal year ending June 30, 1916, the largest average daily output was made at the No. 8 mine of the Old Ben Coal Corporation at West Frankfort, Franklin County. This average was 4865 tons. The mine was operated 193 days. The depth of the shaft is 460 ft. (140 m.).

The total output of this mine was 938,868 tons, and the other of the

two mines then operated by this company produced 743,651 tons. The company has recently acquired four other mines, whose combined production in the same period was 2,474,542 tons.

In some ways, the most impressive record of production is that made at the Superior Coal Co.'s No. 3 mine at Gillespie. This mine produced, in the fiscal year ending June 30, 1916, the largest quantity of coal taken from any one mine in the State, 985,482 tons. The average daily production was 4380 tons. A somewhat better record was made in the calendar year 1916, as the rate of production reached 4403 tons and the total output was 1,023,753 tons in 232½ working days.

Besides this large average daily production and large output, the same mine holds the world's record, so far as can be learned, for production from a shaft mine in a single day. The highest record yet made is 5502 tons in 8 hr., an average of 687.7 tons per hour. This required an average of about 3½ hoists per minute throughout the 8 hr. The depth of the shaft is 340 ft. (103.6 m.). A steam hoist is used.

At the present time this company is opening its No. 4 mine in the same field. It is announced that a daily output of 6000 tons is expected, and there is every reason to expect 7000 tons or more when the operation is fully developed.

The closest rival of Superior No. 3 in point of rapid production is found at the Livingston mine of the New Staunton Coal Co., which is in the same district and is operated under very similar conditions. In the fiscal year ending June 30, 1916, 961,726 tons of coal were produced from this mine, and the average daily output was 4646 tons.

The highest day's output was made on Sept. 30, 1916, when 5287 tons of coal were hoisted in 8 hr. This is an average of 660.8 tons per hour, more than most mines could produce in a day at a time easily remembered by the older persons associated with the industry. There were 1796 hoists, an average of 3.7 hoists per minute. The depth of the shaft is 287 ft. (87.5 m.).

That the coal industry of the State is not in as good condition as is desirable is shown by the fact that the average number of days worked in the fiscal year ending June 30, 1916, according to the Coal Report, was 163. This is hardly a fair statement, for it is the average days worked at all of the mines, and is not a weighted average. If we consider only mines producing more than 100,000 tons in the fiscal year, the number of days of operation was 203.

It is true that the coal industry is a seasonal one to a considerable extent, but not to so great an extent as to warrant an idleness of one-third of the time. If with such a length of idle time the mines produced enough to supply the demand, it is very apparent that there were too many mines and that a smaller number operating more steadily could have supplied the market. The steadier operation of a smaller num-

ber of mines would require the stocking of coal and this subject is likely to receive much more attention in the future than has been given it in the past.

This is one reason for the low profits realized by producers. The cost of operation under such conditions is higher per ton produced than it would be with more steady work, for all the elements of cost, except wages and part of the power and the wear of machinery, continue through idle times. Moreover, the selling price of the coal is made low by the active competition, as the operator naturally desires to keep his mine in operation during as much of the time as possible, and to do this he is inclined to cut the selling price of his coal to the lowest point at which he can make a profit. In many cases this point has been passed and the price has been cut to the cost of production and even to less than this in order that the operation of the mine might be continuous. In some cases this may be good business because it makes possible the maintenance of an efficient working force which might be scattered if the work became too irregular. It would be much better, however, if the operations could be made continuous without sacrifice. These facts have been realized by operators, but they have seen no way to amend the condition without placing themselves in a position to be threatened with prosecution for the violation of Federal laws.

One remedy for this condition of over-production is increase of consumption, and the coal producers of this district have been paying a considerable amount of attention to the increase of markets. Naturally the Illinois coal moves principally to the north and northwest, because the State is environed on the east, south, and west by other coal-producing States. To the east lies Indiana with coals similar to those of Illinois, while farther to the east are Ohio, Pennsylvania, and West Virginia. To the south and east lies Kentucky, whose western coals are similar in type to those of Illinois, while the eastern coals more closely resemble those of the Appalachian region. All of these States ship considerable amounts of coal into Illinois, and even farther to the west and to the northwest.

Illinois is subject to practically no local competition from the west. The Illinois coals, however, have to seek outlet to the west, and in doing so come into competition with the coals of Iowa, Missouri, and Kansas.

To the northwest there is a large field which is a natural outlet for the coals of Illinois. This field, however, was already occupied when Illinois became a large producer, and coals from farther east can reach it at a comparatively low cost because of the advantages of transportation by water over a large part of the distance. This fact explains the interest of coal producers of Ohio, Pennsylvania, West Virginia, and Kentucky, as well as of Illinois, in the cost of transportation into the northwest.

The cost of transportation also has a large part in controlling the movement of Illinois coals toward the west, a reduction in freight rates having made it possible to place some Illinois coals in the market in Kansas City in competition with the Kansas and Missouri coals, which find a natural market in that district.

It is a highly desirable thing for the ultimate welfare of the industry that there should be some control of the markets and the cost of transportation. While the Interstate Commerce Commission properly directs its energies toward the just regulation of freight rates with regard to the cost of the service rendered, it has no right to so control these rates as to lead to the most advantageous development of each coal-producing district. It would be better for the fuel industry if there were some such control in this country as exists in some European countries, through which there should be a proper regulation of output and of markets, so that each district might have its share of the business and at the same time be assured of a fair profit, instead of being forced to surrender almost all, if not all, of that profit in an effort to maintain its position in the industry. I have no doubt that the operators of Illinois, as well as of most other States, would welcome such control, if they could only feel sure that it would be a constructive one.

### *Capacity of Illinois Coal Mines*

An interesting subject at the present time is that of the capacity of the different coal-producing districts. The fact that the mines of Illinois are idle, on an average, during about one-third of the time, shows that the production might be made much larger than it is. The simplest way to get at the possible annual output of the mines is to multiply the average production per working day by 300. An objection to this method is the fact that the average number of days worked is not a weighted average, but is independent of the capacity of the mines. However, if we assume that the average production for the State is about 60,000,000 tons per year and that the mines are operated for 200 days per year, we may say that the capacity of the mines is about 90,000,000 tons per year. This assumption, however, does not allow for the necessity of occasional extensive development work and repairs. It is the custom to assume that there will be a period of low activity during the warm months and to plan for doing some of the development work, construction and repairs during this time. Undoubtedly some of this work could be distributed over the year, but not all of it, and it would be necessary to make some allowance for the time needed for it. This necessity, however, may be balanced against the facts that most mines are not operated at their highest capacity during all of the working period, and that the days counted as working days are probably not all full days. There are no

definite figures available, but it seems safe to say that the mines of Illinois are easily capable of a steady production of not much if any less than 90,000,000 tons per year. The opening of new mines and the better equipment of old ones during the past year have undoubtedly increased the capacity beyond the point indicated by the figures available, but it is not possible at present to say how far.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS  
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## Steam-Shovel Mining of Bituminous Coal\*

BY H. H. STOEK,† B. S., E. M., URBANA, ILL.

(St. Louis Meeting, October, 1917)

### CONDITIONS FAVORABLE TO STEAM-SHOVEL MINING

THE fundamental reasons underlying the choice of a method of mining a coal seam are safety of operation, cheapness of producing the product and the character of the product as a saleable article.

From the standpoint of safety, open or strip coal mining is always preferable, as many of the dangers inherent to underground work are absent, such as explosions of gas and dust, falls of roof and coal, and also to a great extent those due to haulage and the use of explosives, thus eliminating three of the chief causes of accident from which about 75 per cent. of the underground accidents in coal mines occur. The conditions under which coal is stripped also make the work much safer than ordinary quarry work, as the depth of the pit is usually comparatively slight.

From the standpoint of low mining cost, there is much misapprehension and many of the costs given are misleading in that they do not take into account such items as the destruction of the surface for future use, the full cost of the removal of the overburden, depreciation, amortization, coal depletion, loss of time through floods and from bad weather in winter, etc. Coal is no doubt often put on the cars at less cost by stripping than by underground mining, but by no means always so, as is too generally assumed. In some cases coal must be obtained by stripping or not at all, due to the small amount of overburden that renders underground mining impossible.

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\* This paper was prepared at the suggestion of the chairman of the Coal and Coke Committee, and at the outset the writer wishes to acknowledge the coöperation of a number of members of the Institute and others in furnishing much of the data which he has attempted to edit and correlate. Wherever possible due credit is given in the text.

Especial assistance was furnished by Eugene McAuliffe who had begun a paper on the Kansas stripping fields for the St. Louis meeting and who very kindly offered to incorporate his data in the more general paper.

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The cost of opening up a strip mine is unusually heavy. According to J. A. Swanberg of the Carbon Hill Mining Co., the amortization of this charge sometimes amounts to as much as 15 c. per ton as against 3 to 5 c. in many underground operations.

The condition of the marketable product in competition with coal mined by the ordinary methods is a subject that has not received great consideration and is one upon which there will probably be a wide difference of opinion.

C. E. Leshner in the U. S. Geological Survey, *Mineral Resources for 1915*, says:

"Stripped coal does not, however (except locally), bring so high a price in the market as other coal from the same bed, mining from under deeper cover. The average value per ton for the pit coal from Illinois was 10 c. and for Indiana 20 c. per ton less than the average for the State. The difference in Kansas was 8 c.; Missouri, 7 c.; and Ohio, 17 c. The Oklahoma pit coal averaged 7 c. higher than the output of the State."

For the approximate values of pit coal for 1915, see Table 2.

Dr. F. C. Honnold, Secretary of the Illinois Coal Operators' Association and an operator of both stripping and underground operations, suggests that a deduction of 15 c. per ton must be made for the inferior quality of stripped coal because it is obtained near the outcrop. In the Kansas field, Eugene McAuliffe says:

"Coal found under cover of less than 15 ft. (4.5 m.) in thickness is usually soft and low in volatile and sulphur content, high in fixed carbon and moisture, and is not adapted for steam-making purposes, but is consumed by the zinc smelters for fluxing purposes, being sold under the trade name Dead Coal, on account of its inert qualities shown when attempts are made to burn it."

It is the opinion of those connected with the boiler plant of the University of Illinois, where large quantities of Illinois stripped coal have been used, that it requires a larger quantity of this coal to produce a given amount of heat, than of coal from the same region produced by underground mining.

The evidence on this point for other districts is scanty and a discussion of this phase of the subject will be welcomed. If the stripped coal has been subject to weathering, it would naturally be expected to be of slightly lower grade than coal in the same bed found under heavier cover, so that weathering could not take place, but unless weathered, there is no reason why stripped coal should differ from any other coal. In other words, the deterioration is due to the method of occurrence and has no relation to the method of mining.

The conditions under which bituminous coal has been stripped to date are very different from those that obtain in connection with stripping of anthracite; indeed, the conditions have been almost the opposite, for in

the anthracite region the beds have been thick and often steeply inclined, while in the bituminous strippings the beds have been usually flat, are in many cases quite thin, and seldom over 6 ft. (1.8 m.) thick. In connection with anthracite, the dumping area is usually ample and the destruction of the surface for farming purposes does not need to be considered, while some of the bituminous strippings are in very fertile farming country.

The geological conditions that render stripping possible may be briefly given as a bed of sufficient thickness, at not too great a depth, and covered by soft or comparatively soft material. The thickness of coal that is minable is determined by the cost of mining, the quality of the coal, and the price obtainable, which will vary with market conditions. At the present time beds varying from 20 in. to 7 or 8 ft. (0.5 to 2 m.) are being stripped. The thickness of overburden removable depends upon the machinery available and at present the maximum seems to be between



FIG. 1.



FIG. 2.

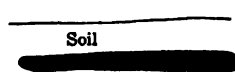


FIG. 3.

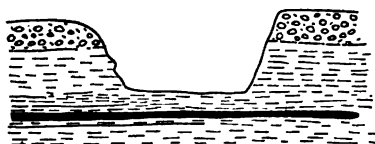


FIG. 4.

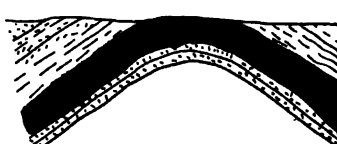


FIG. 5.

FIGS. 1 TO 5.—GEOLOGICAL OCCURRENCES.

40 and 50 ft. (12 and 16.7 m.), but no one can say that steam shovels have reached their maximum development. Comparatively hard shales are easily handled by modern steam shovels, but solid rock, such as limestone occurring in the overburden and requiring blasting, seems to offer a difficulty that in a number of instances has rendered stripping unprofitable.

The geological occurrences of beds that render stripping possible are:

1. A slightly inclined outcropping bed under a comparatively flat surface, Fig. 1. This is the condition in the southeastern Kansas field, where, as noted elsewhere in this paper by Mr. McAuliffe, there are two seams, one varying from 33 to 42 in. (0.8 to 1 m.) in thickness and the other 22 to 25 in. (0.55 to 0.6 m.) in thickness with 65 ft. (19.8 m.) of material between the seams. The dip is small and the cover varies from 8 to 40 ft. (2.5 to 12 m.) of soft slate, shale, clay, and a soft fissured sandstone.

Fig. 2 shows a similar condition where the coal bed may be flat or slightly inclined and where the amount of stripping possible is determined by the slope of the hill.

According to J. H. Sinclair, in the *Colliery Engineer* for June, 1913, there is a bed of coal 10 ft. thick at Tofield, Canada, occurring under a rich prairie loam on a dry sub-soil, which is sometimes from 10 to 15 ft. (3 to 4.5 m.) deep (Fig. 3). The coal area is estimated at 20,000 acres which will give 170,000,000 tons available for stripping.

Fig. 4 shows a condition found in eastern Illinois (Danville district), where a horizontal coal bed was originally overlaid by a considerable thickness of shale and other rock, but small areas have been eroded by streams so as to leave a stripping area surrounded by bluffs.

There are no published records of thick steeply dipping bituminous beds or anticlinal outcrops where stripping is being carried on and such as are common in the anthracite strippings (Fig. 5), but in southern Illinois, near Duquoin, there is said to be a considerable stripping area where the No. 6 coal comes near the surface as a result of the so-called Duquoin anticline, which is more properly a monocline, as only one side is inclined to any extent. A small stripping operation has been carried on in this locality for some years and extensive ones are projected for the near future. The coal is from 6 to 7 ft. thick and occurs under cover varying from 11 to 41 ft.

#### DISTRIBUTION OF BITUMINOUS STRIPPINGS BY STATES AND GEOLOGICAL CONDITIONS THAT HAVE RENDERED STRIPPING POSSIBLE

To determine the area in which stripping is now being carried on, or areas that are available for such stripping, and the geological conditions that render stripping possible, letters were sent to a number of State geologists, heads of mining departments and individual operators, and the following is a digest of the replies received:

##### *Alabama*

E. A. Smith, State Geologist, says:

"A good many years ago this kind of work was done on some considerable scale in Tuscaloosa County, where the conditions were favorable as follows: The occurrence of coal in the Brookwood group high upon the hills with slight cover and the distribution of the group of the Warrior Basin coals may be seen in the Alabama reports on Warrior Basin Coals."

C. H. Nesbitt, Chief Mine Inspector, says:

"There are two stripping operations in Walker County, one located at Carbon Hill on the Jasper seam, being stripped by the Brookside Pratt Mining Co. and another in Walker County, 10 miles (16 km.) north of Jasper, operated by the Sunlight Mining Co. on the Jefferson seam. The geological conditions which render stripping possible are a flat seam with approximately 30 ft. (9 m.) of cover and not much sand rock, the roof being principally slate at Carbon Hill. At Jasper there is sandstone. I can hardly estimate the area being stripped and the coal tonnage available."

The 1915 Mine Inspectors Annual Report gives the output of the Sunlight mine as 1803 tons and 400 tons used at the mines. The mine worked only 40 days during 1915.

J. B. Baddil, of the Sunlight Mining Co., writes on April 17, 1917, that the Sunlight Mining Co. is using a 250-B Bucyrus revolving shovel with a 6-yd. dipper. The company has stripped about 30 acres and has from 7000 to 10,000 acres of coal land available. The thickness of the seam averages 26 in. and the overburden about 25 ft.

### *Colorado*

R. D. George, State Geologist, says:

"There is no stripping of bituminous coal in this State at the present time. A few years ago there was some done in Routt County and a little in Gunnison. I think that was discontinued several years ago. In the instances referred to the coal was overlain by a very much disintegrated shale and in places by stream detritus. None of this stripping was done with a view to production of coal on a large commercial scale, but for local use."

### *Illinois*

There are four different stripping areas in Illinois, as follows:

*Danville District.*—West of Danville, Vermilion County, and near the Indiana border is the district in which so much of the development of the modern steam shovel has been carried on, as noted elsewhere.

The No. 7 coal seam, although usually too deep to be stripped, in this district in a number of instances has an overburden of from 15 to 45 ft. (4.5 to 13.7 m.) due to the fact that streams have eroded valleys and it is in these valleys that the strippings are mainly located. As the coal goes under the hills surrounding these valleys the mining operation is usually a combination of a stripping pit and an underground mine worked by the room and pillar method. The coal is from 5½ to 6 ft. (1.67 to 1.82 m.) thick and yields about 10,000 tons per acre. In this district are a number of brick plants and at a number of them the shale overlying the coal is also mined with the steam shovel and utilized in the manufacture of brick, the underlying coal furnishing fuel for the plant. The brick companies are not generally shippers of coal and mine only for their own use and most of the stripping in the district for coal to be shipped is done by Messrs. Hartshorn, under the several names of the Carbon Hill Coal Co., Missionfield Coal Co., Two Rivers Coal Co. These companies own three shovels, two of which are at present operating, and the third of which will soon be put into operation. These are large Marion shovels, two being type 270, one having a 5-yd. dipper and 90-ft. boom, one an 8-yd. dipper and an 80-ft. boom, and one (type 250)

a 65-ft. boom and  $3\frac{1}{2}$ -yd. dipper. One of these is at present operating in the Two Rivers mine, another at the Carbon Hill mine, and the third at the new Mission mine.

The stripping plant of the Western Brick Co. in the Danville district is of particular interest on account of the variety of operations carried on. The glacial surface till varying from 5 to 10 or more feet in thickness, is first hydraulicked, and then the shale from 20 to 30 ft. (6 to 9 m.) thick is mined with a standard Bucyrus No. 75 steam shovel and used for manufacture of brick, and finally the underlying coal, 5 to 6 ft. (1.5 to 1.8 m.) thick is mined and loaded by hand and used for fuel purposes at the brick plant. The coal is shot loose by charging  $1\frac{1}{2}$  lb. (0.68 kg.) of powder in  $2\frac{1}{2}$ -in. (63.5-mm.) vertical holes, a row of which is drilled 6 to 8 ft. back from the face and about 12 ft. apart. The cost of stripping under these conditions runs up to 40 or 50 c. per ton of coal on the cars.<sup>1</sup>

The Western Brick Co. also has a Marion model No. 250 shovel operating at its No. 3 plant south of Danville.

*Southern Illinois.*—Near Marion, Williamson County, the new Enterprise Coal Co. has about 60 acres (24 ha.) of stripping land covered by  $18\frac{1}{2}$  ft. (5.64 m.) of cover above the No. 6 coal. A Marion 250 shovel with  $3\frac{1}{2}$ -yd. (3.2-m.) dipper and 75-ft. (22.9-m.) boom is used for the stripping and a model 36 shovel with  $1\frac{1}{2}$ -yd. (1.37-m.) dipper is used for loading. The stripping area is an isolated patch of coal and although it is near the outcrop the conditions are not typical outcrop conditions.

Near Duquoin, Perry County, there was for several years a small stripping property worked by a very primitive shovel, which is now not operated. There is, however, considerable stripping activity in connection with lands north and east of Duquoin and in a short time two of the largest type modern shovels will be operating. Messrs. G. W. Dowell and Wm. LaFont of Duquoin estimate that there are some 600 acres in the neighborhood of Duquoin underlain by 7 ft. (2.2 m.) of the No. 6 coal, at a depth of from 11 to 41 ft. (3.4 to 12.5 m.). The outcrop of No. 6 coal westward from Duquoin has been quite thoroughly prospected, according to Messrs. Dowell and LaFont, but no available stripping area has thus far been discovered.

Near Millstadt, east of St. Louis, the Northern Coal Co. had a Marion shovel, type 250—75-ft. boom and  $3\frac{1}{2}$ -yd. dipper. (Now abandoned.)

### *Indiana*

M. Scollard, Mine Inspector, says that there were in Indiana, September, 1916, ten stripping plants, with capacities of 2000 to 20,000 tons

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<sup>1</sup> S. O. Andros: Coal Mining Practice in District VIII (Danville). Illinois Coal Mining Investigation, *Bulletin* 2 (1914), 47.

per month. These plants are described by Mr. Scollard and the operating companies as follows:

Globe Mining Co., near Staunton, mining No. 3 coal 5 to 7 ft. (1.5 to 2.2 m.) thick with 14 to 40 ft. (4.3 to 12 m.) of overburden. Coal is loaded with a small shovel; plant has capacity of 1000 tons per day.

Warren Coal Co., near Hymers, is mining No. 7 seam, 5-ft. (1.5-m.) thick overburden with soil and soft shale. The floor is soft, output is 600 tons per day. One 5-yd. (4.5-m.) Marion shovel is used for overburden and one 2-yd. (1.8-m.) loading shovel for the coal.

Forschner Coal Co., near Linton, is mining No. 4 seam, 5 ft. (1.5-m.) thick with 20 to 30 ft. (6 to 9 m.) of overburden consisting of 18 ft. (5.5 m.) of clay, 12 ft. (3.6 m.) of soft shale and with fire-clay floor. Capacity of 800 tons per day; 60 acres (24 ha.) have been stripped with 50 acres available. A 250 Marion shovel is used, with 3½-yd. (3.2-m.) dipper, and a model 31 is used for loading coal.

Carbon Mining Co. at Carbon is mining the upper Brazil block coal 40 ft. (12 m.) thick, 14 to 25 ft. (4.3 to 7.6 m.) of overburden. In opening the pit, a Class 14 Bucyrus drag line was used and this is now used for hoisting the coal in boxes from the pit; also used for mining 7 to 12 ft. of underlying fire clay. The output of coal is 4000 tons per month. Ten acres have been stripped to date and 20 remain to be stripped. The pit is drained through bore-holes into mine workings 60 to 70 ft. below the strip pit.

J. L. Conover & Co., Carbon, is working block coal 4 to 4½ ft. (1.21 to 1.37 m.) thick with 9 ft. (2.8 m.) of surface clay and 9 ft. of shale overburden. There are 8 to 11 ft. of good fire clay under the coal. The relation between the coal stripping and the fire-clay industry is well described by Mr. Conover as follows:

"At present we are not doing any stripping but are loading the clay from under the coal which we have stripped. We originally started stripping with a 70-C Bucyrus steam shovel, which is a railroad contractor's shovel, and loaded the overburden in Western 4-yd. (3.6-m.) dump cars and hauled the dirt to a spoil dump with 18-ton Davenport dinkies. However, we found this method too expensive. We have contracted the stripping of our coal. The contractor is to strip the coal with a drag line, load the coal on 4-yd. cars in the pit and deliver these cars to the bottom of an incline where they are pulled up to a tippie with a cable and where they dump automatically. We have stripped about 1 acre and have about 10 acres to strip. There are about 20 ft. (6 m.) of overburden, half blue shale and half yellow clay. We grind 7 ft. of fire clay that is found under the coal and ship it to foundries to use in molding and laying brick. Our fire-clay business is by far the best part of our stripping operation."

The Sunlight Coal Co. of Booneville, Ind., is mining the No. 5 seam, 7 ft. (2.2 m.) thick with 20 to 25 ft. of overburden.

The Linton Coal Co. near Linton has one 5-yd. Marion shovel and one small loading shovel. Other companies in Indiana are the Inland

Coal Co. at Janesville, the Richards Coal Co. at Terre Haute and the Eppert Coal Co. at Coal Bluff.

The Linton Fourth Vein Coal Co. at Linton is stripping 12 to 14 ft. (3.6 to 4.3 m.) of shale and clay soil with a 225-B Bucyrus shovel having an 85-ft. (25.9-m.) boom, a 62-ft. (18.9-m.) dipper handle and a 7-yd. (6.4-m.) dipper. This shovel can move 8000 to 10,000 yd. of overburden in two 8-hr. shifts. The coal is loaded into cars with a 35-B Bucyrus shovel which loads from 1100 to 1300 tons per hour.

### *Iowa*

James N. Lees, Assistant State Geologist reports that he does not know of any stripping being done in Iowa.

### *Kansas*

The following account of the southeast Kansas-Missouri area and operations has been furnished by Eugene McAuliffe:

"What is known as the Southeastern Kansas Coal Field is located in Crawford and Cherokee Counties, Kansas, and Barton County, Missouri. Fig. 6 shows the eastern crop line and the western edge of the coal, where the cover is about 260 ft. (79 m.) deep and where erosion has replaced the coal with sand stone. An arrow indicates the dip which is N. 69° 58' W., as calculated by me from levels covering a large number of drill holes.

"Two seams of coal in this field are being stripped (Fig. 7); the lower or Weir City-Pittsburg, varies from 42 in. in thickness in the south end of the field to 33 in. in the north end." (According to H. A. Buehler, State Geologist of Missouri, these coals occur in the Cherokee formation of the lower coal measures.)

"My estimate of the strip coal in this territory and confined to this seam, as originally found in place and now remaining, is substantially as follows:

|  |              |                 |
|--|--------------|-----------------|
| Original area in Kansas.....                                   | 10,320 acres |                 |
| Original area in Missouri.....                                 | 11,680 acres |                 |
| Total.....   |              | 22,000 acres    |
| Removed to date in Kansas.....                                 | 5,920 acres  |                 |
| Removed to date in Missouri.....                               | 2,400 acres  |                 |
| Total.....   |              | 8,320 acres     |
| Remaining to strip.....  |              | 13,680 acres    |
| Tonnage available for stripping Weir City-Pittsburg seam ..... |              | 68,500,080 tons |

"The upper seam, sometimes known as the Lightning Creek seam, while generally present over the entire district, is found of sufficient thickness to warrant stripping only over a limited area. This seam is 65 ft. (19.8 m.) above the Weir City-Pittsburg seam, and where it is now being stripped ranges from 20 to 25 in. (0.5 to 0.6 m.) in thickness. My calculations indicate about 2640 (1068.37 ha.) acres of this seam available for stripping, totaling 8,680,000 tons. In calculating the tonnage of the



lower or Weir City-Pittsburg seam, I have deducted 10 per cent. for faults and horse-backs, which are very numerous in that seam; the Lightning Creek seam being reasonably continuous as to thickness.

"The cover above the coal in the Kansas field ranges from 8 to 40 ft. (2.5 to 12 m.) of soft slate, soft shale and yellow clay. From 5 to 6 ft. (1.5 to 1.8 m.) of soft fissured

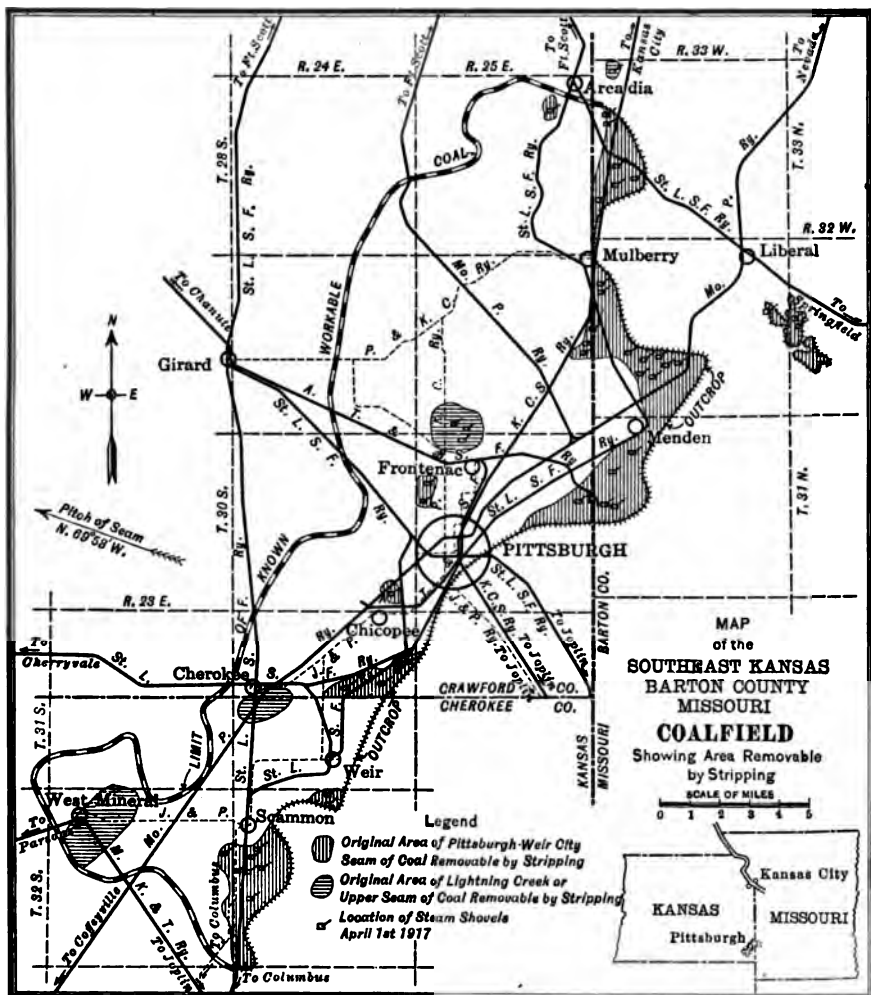


FIG. 6.

sandstone is sometimes found and frequently the shovels move this without blasting, in other cases explosives are required to loosen it. It is not uncommon to strip from 30 to 35 ft. (9 to 10.6 m.) above the coal.

"In the northeast corner of the map in the hatched area there are three shovels working; the coal seam going back under a flat hill. There are 45 shovels now working in the southeastern Kansas field, the majority being of a modern type. Some of these shovels have a 90-ft. (27.5-m.) boom and the dipper handles vary from 45 to 55 ft.

(13 to 16 m.) in length, the dipper in some cases being  $6\frac{1}{2}$  yd. (5.9 m.) capacity. One shovel is working at Browington, Henry County, Missouri, but the area of stripping coal is there very limited."

### *Kentucky*

J. B. Hoeing, State Geologist, reports that he knows of no stripping being done in Kentucky at present. Some was done in Laurel County years ago, as is noted later under the development of the steam shovel.

### *Michigan*

R. C. Allen, Director, Michigan Geological and Biological Survey, reports that no coal is being stripped in Michigan at present. A small amount of coal was stripped in Williamston, Ingham County, about 2

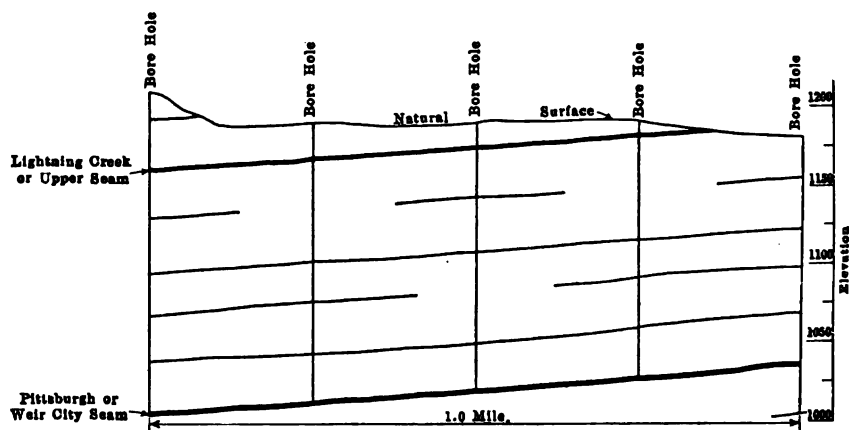


FIG. 7.—SEAMS STRIPPED IN KANSAS.

years ago, but on a very small scale. At Grand Ledge, Eaton County, a small amount of coal is obtained in connection with the winning of clay. Excepting for these operations at Williamston and Grand Ledge, the conditions in Michigan are not favorable for stripping, as the coal basin is generally deeply covered by drift and the coal is from 50 to 250 ft. (15 to 76 m.) in depth.

### *Ohio*

J. A. Bownocker, State Geologist of Ohio, reports that stripping is being carried on in an extensive way in the No. 8 or Pittsburgh coal in eastern Ohio. There are not more than a half dozen plants in operation at the present time, since the country is so hilly that the areas where stripping is possible are not large. There is also a plant on the border of Stark and Tuscarawas Counties on which the lower Kittanning bed is

being stripped. Ohio has about a dozen coal beds that are being mined which are persistent across the southeastern part of the State. At many places they may be mined by stripping and the areas though large would form only a small part of the entire coal field. No figures are available giving the amount of coal obtained by stripping.

S. A. Taylor, of Pittsburgh, who is interested in Ohio stripping propositions, says that the location of the Ohio stripping is at present in Harrison and Jefferson Counties in eastern Ohio in the No. 8 field largely. There are five or six different companies operating in this district. The property of the Kehota company with which he is connected is located in Perry County in the Hocking district. He says:

"We have tested quite a number of properties in Hocking and Perry Counties, as well as in others, but have not found any available, excepting two in Perry County. It is possible that others will be discovered by drills now working."

It is rumored that a number of shovels have been purchased recently to operate in Ohio.

### *Pennsylvania*

In regard to southwestern Pennsylvania, S. A. Taylor says:

"There is some coal in western Pennsylvania along the Ohio River, bordering West Virginia, which may be stripped. While the overburden is light, it is composed largely of rock and I fear that the cost of stripping may be prohibitive. An attempt to strip in this section will soon be started by John Bell and George Flynn of Pittsburgh. As to stripping the Upper Freeport coal, I do not think this is feasible, as this coal is capped by Mahoning sandstone which is usually a massive sandstone of considerable thickness and unless locations were found where the rock was eroded, leaving the thin black slate immediately over the coal, I do not think it would be a commercial stripping proposition."

Richard R. Hice, State Geologist, says:

"I think there is quite a little area of the Pittsburgh bed in Washington and Allegheny Counties that can be mined by stripping and, indeed, only mined in this manner. At present most of this is too distant from transportation. The possibility of stripping this coal is due to the fact that the general rise of the rocks to the north has brought the Pittsburgh coal to daylight. It is now found close to the top of the hills with but little cover. The area lies outside of the glaciated region. It is possible that the Pittsburgh bed can be stripped farther east and I know of no reason why there should not be places where considerable areas of the Allegheny coals might also be mined in this manner, but we have no exact information on this subject. I understand there is a prospect of mining some Upper Freeport coal by stripping."

Joseph Knapper, Mine Inspector, Philipsburg, Pennsylvania, reports that at Karthous an operation belonging to the Black Diamond Coal Co. owned by Rembrandt Peale was abandoned by some Italian miners who contracted to deliver coal from about 10 acres at a price which should compete with pick mining. They worked for about 6 weeks, removing the cover, which varies from a few inches to 6 ft. (1.8 m.) and used only

pick and shovel. At that time they had uncovered about 200 tons of coal, but most of it was smut which was not fit for use. They attempted to collect days' wages from the coal company, but the suit that was brought was soon dropped, and the attempt at stripping ceased.

### *Oklahoma*

C. W. Shannon, Director, Oklahoma Geological Survey, says:

"Considerable stripping has been done near Coalgate and Lehigh in the southern part of the field, but the greater part of this has been done by teams and consequently no very large quantity has been removed. Some of the companies have been considering the steam-shovel proposition in the outcrops of these districts. However, this has not been undertaken. The coal of this vicinity is from 3 to 4 ft. (0.9 to 1.2 m.) thick and the overburden is of such character that it can readily be removed by the use of a steam shovel. Other districts in the State in which coal is being removed by stripping are near Collinsville, in Rogers County, and near Broken Arrow in Tulsa County, the mining, however, being chiefly across the county line in Wagoner County.

"Practically all of the coal in the State lies at a considerable angle and of course outcrops over long and sinuous courses."

*Bulletin 22* of the Oklahoma Geological Survey, page 72, under the heading "Mining Coal by Strip Pits" says:

"In northeastern Oklahoma and southeastern Kansas there are several thin beds of coal covering large areas. Until very recently these coals had not received very favorable notice. These deposits consist of almost continuous beds and are from 1 ft. 6 in. to 2 ft. (0.46 to 0.6 m.) in thickness. The coal is of extra good quality for shallow coal. These beds are covered with an overburden of shale and clay from less than 10 to 30 ft. (3 to 9 m.) in thickness. Some of these deposits, as well as narrower strips along the outcrop of the principal beds in the main part of the field, have been worked in the past by stripping with horses and scrapers, but this method was not found practicable where the covering is more than 10 ft. (3 m.). During the past few years steam-shovel operations have begun in these areas and a marked development has taken place. Two shovels of the largest type are operating in the Oklahoma field, one near Lehigh and the other at Henryetta."

*The Handbook of the Natural Resources of Oklahoma*, page 13, says:

"In the northern end of the coal field are several beds of coal averaging from 1 ft. 6 in., to 3 ft. 6 in. (0.46 to 1.06 m.). These coals dip but slightly and are not far from the surface. Mining operations are being carried on in these coals at Bluejacket, Catale, Collinsville, Dawson, Tulsa, Broken Arrow, and Pryor. The coal mined is of a good quality, but is softer than the deeper coals. At Collinsville and Broken Arrow the coal is being stripped with a steam shovel and this method of securing coal has proved to be very satisfactory.

"There are many miles along the outcrops in various parts of the field where a little coal has been stripped in former years. The stripping was formerly done by teams. There are large areas throughout the coal region where steam-shovel stripping may be successfully carried on. The thickness of the surface which can be economically removed depends on the thickness and quality of the coal, but in most cases 25 to 50 ft. (7.6 to 15 m.) can be removed."

*Tennessee*

A. H. Purdue, State Geologist, reports that there is no coal mined by stripping in Tennessee.

*West Virginia*

I. C. White, State Geologist, reports that so far as he knows there are no stripping mines in that State, due to its very hilly character.

## DEVELOPMENT OF THE STEAM SHOVEL FOR STRIP MINING

In looking into the history of steam-shovel mining, it was found that Grant Holmes of Danville, Ill., for many years closely associated with the manufacture of steam shovels and with the development of the Mission Field Mining Co. near Danville, had gathered a large amount of data upon the development of the steam shovel in connection with coal mining. These data he has most kindly given permission to have incorporated into the present paper and, for practically all of the following historical section, credit is due Mr. Holmes.

In 1866, Messrs. Kirkland, Blankeney, and Graves opened a strip mine at Grape Creek south of Danville, Ill. In 1875, the late Michael Kelly opened a similar mine at Hungry Hollow, also near Danville, and a number of similar operations were begun at about the same time in this same district. The coal was usually exposed in long pits, a section along one edge of the field being first plowed, the surface removed with scrapers, and the overburden piled in a long mound overlooking the pit. This work was usually done in the summer and the coal was mined and hauled out during the winter by wagons. The following summer a block parallel to the one previously mined out was stripped, the waste being dumped into the abandoned cut. In 1876 and 1877, J. N. Hodges and A. J. Armil began stripping near Pittsburg, Kan., depending upon the experience of Mr. Hodges in using steam shovels for building railroads in Ohio. They decided, after stripping coal with teams and scrapers for a while, that the work would be better done with a steam shovel. In 1877 they rented an Otis steam shovel and this is the first record of steam-shovel coal stripping. In regard to this pioneer machine, Mr. Hodges says:

"We operated the shovel for about 1 year very successfully and I sold my interests to engage in other business, but the shovel was operated very successfully for about 3 years, when it was returned to the owners. We had land with from 8 to 12 ft., (2.4 to 3.6 m.) overburden. This with good management could be handled very well, but coal at this depth was limited and the boom of the shovel was too short to do deeper work and waste the overburden far enough away to uncover a pit of coal of sufficient width to handle economically." Mr. Hodges expressed the opinion that a larger shovel could be successfully used for such work, but the manufacturers said that such a shovel was not practicable.

The second steam-shovel experience was that of the Consolidated Coal Co. of St. Louis, which owned a stripping proposition in the river bottom, known as Mission field, near Danville, Ill. The overburden did not exceed 35 ft. (10.6 m.) and was often only 10 to 15 ft. (3 to 4.5 m.) deep, while the coal bed was 6 ft. (1.8 m.) thick. Wright and Wallace of Lafayette, Ind., dredge contractors, accepted the contract to uncover a fixed amount of coal daily for the Consolidated Coal Co., under the supervision of J. L. Swansberg and Louis Stockett of the Consolidated Coal Co. They accepted the opinion of Hodges and Armil, that steam shovels had not been developed to a size suitable for stripping coal economically and, being dredgemen, Wright and Wallace purchased from the Marion Steam Shovel Co. of Marion, Ohio, a dredge minus the hull, which they erected on a wooden frame (Fig. 8), supported on wheels, thus forming a dry-land dredge built entirely of wood. The boom was 50 ft. (15 m.) long and a single-cylinder vertical steam engine furnished



FIG. 8.—WRIGHT AND WALLACE SHOVEL.



FIG. 9.—DREDGE AND STACKER.

power to hoist the  $\frac{3}{4}$ -yd. (0.68-m.) dipper and to swing the boom. With this machine, 400 cu. yd. (305 cu. m.) of overburden per day was a great record. The only means of propelling the dredge was by block and tackle; hence the moving of the outfit was a slow process, particularly on curves, as the wheels were rigidly fastened to the frame. This dredge moved forward in an elliptical spiral path and, as the curves became short, jacks were used to skid the dredge around the sharp curves. The widest cuts possible with the dredge were 20 ft., and on account of the limited dumping range waste banks could not be placed far enough away to prevent the dirt from rolling down and covering up the face of the coal. Because of this gradual covering of the coal, it was necessary for the coal miners to follow immediately after the stripper and also to cut an entry in the coal before it could be loaded. The slowness of operation, the narrowness of the cut, and the annual flooding of the field from the neighboring stream induced the contractors to place a second shovel

in an effort to keep their agreement with the Consolidated Coal Co. This second dredge or shovel was similar to the original one, but larger, having a  $1\frac{1}{4}$ -yd. (0.38-m.) dipper. Soon after, a third similar machine was purchased with a  $1\frac{1}{2}$  yd. (0.46-m.) dipper, 65-ft. (19.8-m.) boom, and two vertical steam engines. In 1888, an unforeseen complication arose in the nature of a strike of the coal miners, thus shutting down the coal-producing part of the operation. As this contingency had not been provided for in the contract, the stripping contractors continued to operate the dredges or shovels during the strike, and a point was soon reached where the shovels had made a complete round of the workings and were starting on a second cut, depositing the overburden upon the coal which had been stripped on the preceding round. To overcome this, the coal company bought the dredges and the contract, and when the strike was finally settled the stripping work was resumed and the three machines operated for 2 years by the Consolidated Coal Co. Apparently no money was made and owing to the heavy expense and the slow rate of operation and to the fact that machines were about worn out, not having been built to stand such heavy work, and after many fruitless attempts to repair them, in 1890 they were abandoned.

The next development in steam-shovel mining was an adaptation of a scraper line used in the production of clay for ballast burning in Kansas and Missouri. This gumbo or sticky clay was burned in trenches in which alternating layers of coal and clay were placed. In the digging of the trenches the Butler Brothers (Henry G. and William) who were the ballast burners for the Rock Island Railroad and who previously came from Kenosha, Wis., used drag lines and a cable-manipulated scraper bucket.

The Consolidated Coal Co., after investigating this drag-line operation, made a contract with Butler Brothers to finish the stripping at Mission field. They began work in 1890 with three drag lines with bucket capacities of  $\frac{3}{4}$ ,  $\frac{7}{8}$  and 1 yd. respectively. The drag or scraper buckets were steel boxes with one open end and with teeth on the bottom edge. A cable passing through a sheave wheel on the top of drag served as an adjustable trolley on which the buckets moved forward and backward and by means of which it was raised and lowered. By means of ropes fastened to the two ends, the buckets were moved backward and forward and by means of another cable the latch was tripped so that the bucket swung downward to dump the load. These various cables were strung along a horizontal boom 80 ft. long, which boom was suspended from a vertical frame or gantry by ropes. Two vertical boilers supplied power for a two-cylinder engine geared to three drums which were controlled by clutches. The machine was self-propelled in either direction by means of gearing operated from the same engine.

These drag lines worked on top of the bank and near the edge the

horizontal boom extended over the cut. The machines worked with accuracy and speed and three dumps per minute was not an unusual record. As the boom could not swing, the whole machine was moved to keep the scraper supplied with digging material, and this required much track-laying of an expensive nature to prevent the drag line from sinking into the soft overburden. With these drag lines, a rectangular cut was made about 20 ft. (6 m.) wide and a  $\frac{1}{4}$  mile (0.4 km.) long, the boom being of sufficient length to permit the waste bank to be built far enough from the coal face to prevent burying the face by sliding spoil. When the end of the cut was reached, the machine was moved back about 20 ft. from the edge of the bank and then was worked back in the reverse direction from the previous cut, the spoil being deposited in the cut from which the coal had meanwhile been quarried. These first operated on what is known as the upper bottom of Mission field, where Wright and Wallace had previously operated. The stripping was shallow and the overburden contained no hard material, so that the operation was carried on with ease and speed and an output of 1000 tons of coal a day was obtained. The available easy stripping ground in this section did not last very long, as much of the coal had been mined by Wright and Wallace in their previous work, and because of increased overburden, unexpected appearance of hard blue shale above the coal, and bad floods, the Butler Brothers<sup>2</sup> moved two of the excavators to what is known as lower Mission field, the third machine being abandoned. In this new field, the overburden was light and rapid work was made on a 40-acre tract of coal. The machines operated in tandem, one stripping, and the other, loading the coal directly into railrod cars which ran into the pit, making one of the first examples of machine coal mining. In the center of the field, however, shale and sandstone were encountered and the depth of overburden increased. The work progressed more slowly and it became necessary to drill and blast the hard material, thus increasing the cost of production. Butler Brothers' contract with the coal company was for a certain price per ton delivered on coal cars and, as the result of the difficulties in digging and the trouble with the miners who struck for an 8-hr. day and increased wages, the drag lines were abandoned.

Butler Brothers still believed in this type of excavator and thought that a larger one would be successful. Financed by the Consolidated Coal Co., they began the erection of a large drag line which should handle deep stripping, shale, soapstone and coal and in 1900 this large machine was completed at a cost of \$30,000. It was supported on three 10-in. (0.25-m.) axles, each 22 ft. (6.7 m.) long, the center axle being geared to the engine and the end axles connected to the center one by driving rods. The horizontal boom was 135 ft. (41 m.) long and was supported by a vertical gantry 60 ft. (18 m.) high. To balance this, the rear end of the

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<sup>2</sup> H. H. Stock: Dredging for Coal. *Mines and Minerals* (August, 1901), 5.



machine was loaded with 25 tons of iron, but later, as the result of a fire, the horizontal boom was shortened by 20 ft. (6 m.) and then the difficulty was to keep the machine from tipping backward. Three buckets were used, each holding about 2 cu. yd. (1.5 cu. m.), one intended for dirt, one for rock and the third for coal. The difficulty in operating this machine was its immense weight which caused both rails and ties to be buried in the ground, especially those near the edge of the bank, with the resultant constant fear that the whole machine would slide into the pit. The rock bucket, which was later fitted with a knife to cut into the hard material, was not successful, as it could not be held into the cut, and drilling and blasting, at a heavy expense, was necessary. The loading of the coal with the machine was entirely successful and 25 gondola cars could be filled a day. After 2 years of unsuccessful effort to keep to their part of the contract, Butler Brothers gave up the undertaking, but 16 of their employees leased the equipment and, as a coöperative venture, under the name of the Salt Fork Co., operated with fair success in a section where the coal was not covered with hard material. In 1904 this enterprise failed, due partly to there being too many coöperators.

In 1903, Mr. Donovan and associates with a Schenable drag line operated in the Middle Fork bottom of Mission field in a neighboring valley to the one the Salt Fork Co. was operating. This drag line, though much smaller than those of the Butler Brothers, revolved in a complete circle, and for 9 months this machine operated, but, due to overflow from the adjoining river, the waste banks slid down so badly that no coal could be taken out. The outfit and property were bought by Edward Gray who organized the Gray Coal Co. but after 5 months of unsuccessful operation he also gave up.

Mr. Gray was next made manager of the Salt Fork Coöperative Co., but in less than a year's time the coöperators again took over the proposition into their own hands and for 2 years attempted to carry on the work. The occurrence of the hard soapstone, excessive repair bills and floods from the nearby river, resulted in the failure of this enterprise for the third time. Thus, after a period of 9 years and as a result of the efforts listed, only 7 or 8 acres (2.8 to 3.2 ha.) had been stripped, at a cost of approximately \$100,000.

In 1907, under conditions similar to those in the Mission field, a stripping proposition was started near Lilly, Ky., in the Robinson Creek bottom where Jellicoe coal was 28 in. (0.71 m.) thick with only 6 to 8 ft. (1.8 to 2.5 m.) of overburden. Two shovels were used, one for stripping, the other for mining, built by the Vulcan Steam Shovel Co. of Toledo, Ohio. The stripping shovel had a  $2\frac{1}{2}$ -yd. (2.3 m.) dipper and a 28-ft. (8.5-m.) boom mounted on railroad trucks. The mining shovel known as the Vulcan "Little Giant" had a  $1\frac{1}{4}$ -yd. (1.2-m.) dipper, a 22-ft. (6.7-m.) boom and was mounted on traction wheels. After 2 years'

trial the work was discontinued, the principal trouble being that water caused the waste banks to slide into the pit.

The Standard Block Coal Co. tried to operate a Monighan drag line in this district, but unsuccessfully, and after 1 years' work the attempt was given up.

Thus, in both Mission field and Robinson Creek bottom, steam shovels were first used and then drag lines. A combination of these two types was tried by the Danville Brick Co. of Danville, Ill. in a tract adjacent to its brick yards, which contained about 4 acres of coal under 25 ft. (7.6 m.) of cover. A small Vulcan steam shovel worked on top of the coal and dumped the spoil at one side. Moving parallel with the shovel, a small drag line mounted on top of the nearest waste bank picked up the spoil from the shovel and deposited it in front of the drag line, thus laying the foundation for its own tracks and also building the waste bank. As this coal was intended for use in the brick plant of the operating company, the question of reasonable profit did not enter in so long as the coal could be produced below the current market prices. This piece of coal was uncovered during the fall and winter of 1912-13.

About 6 miles southeast of the Mission field is a stripping area called the Carbon Hill district, containing about 70 acres (28 ha.) underlaid by No. 7 coal 6 ft. (1.8 m.) thick, with an overburden of about 40 ft. (12 m.) of sand, gravel, loam and a heavy bed of shale. This tract was purchased by the Consumers Coal Co. of Danville in 1904, who contracted with George W. Prutsman, an excavation contractor, to strip the area. The machine used was built by the Bellefontaine Foundry & Machine Co. of Bellefontaine, Ohio, according to the joint ideas of Mr. Prutsman and George E. Turner, who were familiar with the failures of the drag line. They adopted the steam shovel, but instead of using a long-boom shovel, as Wright and Wallace had done, they incorporated a belt conveyor to handle the spoil dug with a short-boom shovel. This shovel could dig rapidly and deposit the waste in a hooper (Fig. 9) that fed the conveyor belt (Fig. 10), operating at right angles to the direction of digging. The shovel was operated by four two-cylinder engines, one for hoisting the 2-yd. (1.8-m.) dipper, a second for crowding it, a third for swinging a 35-ft. (10.6-m.) boom, and the fourth for operating the conveyor, which was 105 ft. (32 m.) long. The machine was built of wood and was mounted on four four-wheel trucks, thus giving a four-point suspension, no attempt being made to keep the frame level. It was propelled by block and tackle. The machine could only dig forward, hence the circular plan of operation was used. The work was begun in 1904, the surface being stripped in the summer and coal loaded out in winter. After several attempts to get the field in shape for circular operation, the plan was abandoned on account of the irregular shape of the coal area and the great depth of overburden in certain places. To overcome

this the work was opened in parallel cuts, which necessitated leaving the coal in the cut after one cut of the spoil had been completed, until the excavator could be moved back to the starting point for a second cut. This moving back usually took 2 weeks and the parallel cuts were not straight but had many inside turns in them; that is, turns in which the dumping arc is smaller than the digging arc. The end of the conveyor boom was thus practically a center about which the machine traveled during these turns and consequently the waste bank would build up so as to clog the conveyor. The conveyor was lengthened to 147 ft. (44.8 m.) but this did not obviate the difficulty; the bills for repairs were heavy, especially for the conveyor belts, and after 2 years, Mr. Prutsman gave up the contract. Backed by other men who formed the Coal Production Co., he took a new contract which in a few months also stopped for lack of funds. The Consumers Coal Co. then decided to



FIG. 10.—STACKER AND SPOIL BANK.



FIG. 11.—DRAG LINE LOADING COAL.

operate the stripping itself and the shovel was completely rebuilt, steel being substituted for wood. However, in spite of a better shovel and careful management, in 1913 this company quit business without having made a profit. This type of machine was therefore abandoned as a theoretical success but as a practical failure.

The combination of shovel for digging and conveyor for depositing of the spoil seems to be a good one. Morton E. Pugh, who spent a fortune and lifetime with conveying-belt strippers, took out his first patents on a machine that moved and dug sideways, the conveyor dumping behind. His later patents show that he separated the excavator and the conveyor into two independent machines, the steam shovel being substituted for his excavator and a crusher being added to the conveyor for sizing the waste before dropping it on the belt, the two machines being operated on parallel tracks. Mr. Pugh also planned to use a revolving steam shovel, so that the conveyor could be on the same track behind the shovel, but several attempts at such operation made in Iowa were unsuccessful.

The Carbon Hill stripping area is now being operated by the Carbon Hill Coal Co., with a revolving 5-yd. (4.5-m.) dipper shovel.

### *The Revolving Stripping Shovel*

In 1909, the Missionfield Coal Co., of which W. G. Hartshorn was president, leased the stripping equipment described above as belonging to the Consolidated Coal Co., and after 2 months' trial operation bought the property with the equipment described. Easy stripping, the absence of floods and labor trouble, permitted a good profit to be made with the drag line for about 16 months, but then repairs became excessive and the equipment was abandoned. Those connected with the Missionfield Coal Co., although recognizing the faults in the Prutsman machine, believed that the ultimate stripping machine must combine the dumping range and the digging power over the conveyor steam shovel with an ability to dig and dump in any direction; that is, it should be a revolving shovel. They believed that the appliances built along these lines had failed through being too small to stand up to the work, and that the new type of stripping machine must be of greatly increased size, but although several of the steam-shovel manufacturers were already making small revolving shovels and locomotive cranes, none seemed willing to undertake the construction of a large revolving shovel.

In 1910, the Vulcan company built two revolving shovels for stripping purposes with  $1\frac{1}{2}$  yd. (1.37-m.) dippers and with booms about 50 ft. (15 m.) long. Messrs. Patrick Dirky and Joseph Stevenson each purchased one of these machines for use near Pittsburg, Kan. The shovels were badly proportioned in the swinging parts and constant breakdowns resulted. While these original shovels have been operated by a number of different men since that time, and are still being operated, they have never been very successful.

The Brown company of Cleveland, Ohio, next modified a locomotive crane so as to form a revolving steam shovel with a 2-yd. (1.8-m.) dipper and a 50-ft. boom, and in 1910 this machine was put to work in the Mission field. In 6 months the remains were sent back to the factory, as the machine was too lightly constructed, but it was evident that a start had been made in the right direction and after 2 years of argument, the Marion Steam Shovel Co. began the construction of a revolving shovel with the dimensions and according to the ideas of Messrs. Grant Holmes and W. G. Hartshorn of Danville, Ill. The machine had a  $3\frac{1}{2}$ -yd. (3.2-m.) dipper, 40-ft. (12-m.) handle, 65-ft. (19.8-m.) boom, and weighed 150 tons, making the largest shovel in the world up to that time. A unique feature was the hydraulic compensating truck which was patented by Mr. Holmes and which is now used on all shovels of the Marion type, by means of which the shovel frame is kept level when the apparatus is

moved over irregular tracks. In 1911, this big shovel, known as Model 250, began work in the Mission field, where it continued until the field was stripped completely, toward the latter part of the fall of 1915, thus completing the work of stripping the field where the early experiments of the Butlers and Brownings had been carried on. After 4 years of successful and profitable operation, the shovel was in good condition and repairs had been very slight. From the Mission field it was moved to an adjoining stripping basin owned by the same interests, where it is still in operation. The operation of this shovel was watched with interest and, as soon as it was a demonstrated success, others were built. The large-size revolving shovel has now become the feature in coal-stripping operations.

Drag lines are used in a few places and their general construction is similar to that of the steam shovel in so far as the operating devices are concerned.

### *Steam-Shovel Construction*

A steam shovel should be able to work under severe conditions, as nearly as possible continuously, and with a minimum of breakdowns, as any shovel delay usually means a delay along the entire line of operation. The principal points to be considered in comparing shovels are:

|  |                    |
|--|--------------------|
| Pounds of pull at the dipper.                | Ease of movement.  |
| Height of the lift.                          | Repairs.           |
| Distance at which material can be deposited. | Cost of operation. |
| Capacity of bucket.                          | Cost of repairs.   |
| Weight of shovel.                            | Investment cost.   |

The largest shovels used in stripping for bituminous coal are either of the Marion model 271 or 300, or the Bucyrus 150-B, 175-B, 225-B. Figs. 12 and 13 show photographs of the latest of each of the largest types. These are described in great detail in the excellent catalogs of the two companies noted and only a few distinguishing points need here be given. They are designed to strip overburden from approximately horizontal beds of coal and other deposits where a great reach is needed, so that the spoil need not be hauled away in cars, but can be deposited either upon the original ground level or in the excavation from which the coal has been removed. As the entire superstructure revolves the shovel can work back and forth along the face by simply shifting the track and without turning the shovel. The booms vary from 60 to 90 ft. (18 to 27 m.) in length<sup>3</sup> and the dippers from 2½ to 8 yd. (2.29 to 7.3

<sup>3</sup> 8-yd. dipper Marion has an 80-ft. boom.

6-yd. dipper Marion has an 90-ft. boom.

150-B Bucyrus has an 60-ft. boom.

225-B Bucyrus has an 85-ft. boom.

m.). For a given model, the larger the dipper the shorter the boom, to prevent overturning. In order to keep the platform horizontal when the shovel is moved, the Marion shovel is supported upon a compensating



FIG. 12.—MARION SHOVEL, MODEL 300, PITTSBURG, KANS.



FIG. 13.—BUCYRUS 225-B SHOVEL, CARNEY CHEROKEE COAL CO., KANSAS.

hydraulic device patented by Mr. Holmes of Danville, and with the Bucyrus shovel the same end is accomplished by a three-point support with screw jacks. The Bucyrus shovels have a split dipper handle which

straddles the boom (Fig. 17), but the Marion has the dipper handle solid and working between the two parts of the boom. The later types of shovel have a very heavy built-up steel boom. The shovels are either operated by steam or electricity and the makers rate the capacity of the larger sizes as from 1500 to 5000 cu. yd. (1147 to 3823 cu. m.) per 9 hr. The larger shovels remove an overburden up to 45 to 50 ft. (13.7 to 15 m.) deep and make a cut from 40 to 110 ft. (12 to 33.5 m.) wide, depending on the depth of overburden, length of boom, and length of dipper handle. The width of cut varies inversely with the depth in order to provide suitable space for the waste. The catalogs of the several shovel makers contain tables of relative depths and widths of cut of shovels of different dimensions.

Shovels should be able to load cars standing on a track not less than 6 ft. above the shovel track and to do this the distance from the shovel track rail to the lowest point of the dipper door when open should be at least 14 ft. or more for advantageous working.

After material has been loosened by the steam shovel, it occupies from

TABLE 1

| Company<br>Manufactur-<br>ing Shovel | Type  | Boom | Handle | Dipper | Where Operated and Number |      |      |      |     |      |       |    | Total |
|--------------------------------------|-------|------|--------|--------|---------------------------|------|------|------|-----|------|-------|----|-------|
|                                      |       |      |        |        | Ala.                      | Ill. | Ind. | Kan. | Mo. | Ohio | Okla. |    |       |
| Bucyrus....                          | 150-B | 68   | 38     | 2½     | 1                         | ...  | 1    | 5    | 1   | ...  | ...   | 8  |       |
| Bucyrus....                          | 175-B | 75   | 48     | 3½     | ...                       | ...  | ...  | 8    | 1   | 1    | ...   | 10 |       |
| Bucyrus....                          | 175-B | 85   | 58     | 3½     | ...                       | ...  | 1    | ...  | ... | ...  | ...   | 1  |       |
| Bucyrus....                          | 225-B | 80   | 58     | 6      | 1                         | ...  | 2    | 3    | 1   | 5    | ...   | 12 |       |
|                                      |       | 75   | 58     | 6      | ...                       | ...  | ...  | 2    | ... | ...  | 1     | 3  |       |
|                                      |       | 80   | 58     | 7      | ...                       | ...  | ...  | 1    | ... | ...  | ...   | 1  |       |
|                                      |       | 80   | 54     | 7      | ...                       | ...  | ...  | 5    | ... | 3    | ...   | 8  |       |
| Marion....                           | 211   | 45   | ...    | 2½     | ...                       | ...  | 1    | ...  | ... | ...  | ...   | 1  |       |
| Marion....                           | 250   | 65   | ...    | 3½     | ...                       | 1    | 2    | 7    | 1   | ...  | ...   | 11 |       |
| Marion....                           | 250   | 75   | ...    | 3½     | ...                       | 2    | 1    | 1    | 2   | ...  | ...   | 6  |       |
| Marion....                           | 251   | 75   | ...    | 3½     | ...                       | ...  | 1    | 2    | 2   | ...  | ...   | 5  |       |
| Marion....                           | 251   | 75   | ...    | 4      | ...                       | ...  | ...  | ...  | 1   | ...  | ...   | 1  |       |
| Marion....                           | 252   | 75   | ...    | 3½     | ...                       | ...  | ...  | 1    | ... | ...  | ...   | 1  |       |
| Marion....                           | 270   | 90   | ...    | 5      | ...                       | 1    | ...  | 2    | ... | ...  | ...   | 3  |       |
| Marion....                           | 270   | 80   | ...    | 8      | ...                       | 1    | 1    | ...  | ... | 1    | ...   | 3  |       |
| Marion....                           | 271   | 90   | ...    | 5      | ...                       | ...  | 3    | 1    | 1   | 1    | ...   | 6  |       |
| Marion....                           | 271*  | 90   | ...    | 6      | ...                       | ...  | ...  | ...  | ... | 1    | ...   | 1  |       |
| Marion....                           | 271   | 80   | ...    | 8      | ...                       | ...  | ...  | 1    | ... | ...  | ...   | 1  |       |
| Marion....                           | 300*  | 80   | ...    | 8      | ...                       | ...  | 2    | 1    | 2   | 2    | ...   | 7  |       |
| Marion....                           | 300*  | 90   | ...    | 6      | ...                       | ...  | ...  | ...  | ... | 3    | ...   | 3  |       |
|                                      |       |      |        |        | 2                         | 5    | 15   | 40   | 12  | 17   | 1     | 92 |       |

\* Electrically operated.

25 to 50 per cent. more space than when solid and the waste bank will take up considerably more space than the original solid bank. To provide ample space for mining the coal, the bottom of the waste bank should be kept at least 10 ft. away from the coal face. If part of the overburden dug by the shovel is harder than the main portion and therefore does not run as readily as the main spoil bank, this harder material may be deposited nearest the coal face and thus serve as a retaining wall to hold back the more easily running spoil. Table 1, based mainly on lists furnished by the Bucyrus and Marion companies, shows the distribution of the steam shovels in the United States, July 1, 1917.

The statistics of steam-shovel operations are somewhat meager. A summary of those collected by the United States Geological Survey for 1915 is shown in Table 2. Data regarding the depth of cover and yardage handled were published in the report of this series for 1914.

TABLE 2.—*Coal Recovered from Steam-Shovel Strip Pits in 1915*

| State                                 | Number of Shovels | Quantity of Coal Mined (Net Tons) | Average Tonnage per Man |          | Average Value per Ton |
|---------------------------------------|-------------------|-----------------------------------|-------------------------|----------|-----------------------|
|                                       |                   |                                   | Per Day                 | Per Year |                       |
| Alabama.....                          | 1                 | (a)                               | ....                    | .....    | .....                 |
| Illinois.....                         | 10                | 455,195                           | 8.7                     | 1,979    | \$1.00                |
| Indiana.....                          | 22                | 638,220                           | 8.3                     | 1,474    | 0.90                  |
| Kansas.....                           | 23                | 780,787                           | 5.3                     | 1,169    | 1.58                  |
| Missouri.....                         | 20                | 655,670                           | 4.4                     | 870      | 1.66                  |
| Ohio.....                             | 7                 | 273,263                           | 7.0                     | 1,763    | 0.91                  |
| Oklahoma.....                         | 4                 | 28,484                            | 4.3                     | 274      | 2.08                  |
| Total bituminous (except Alabama).... | 87                | 2,831,619                         | 6.0                     | 1,208    | 1.29                  |
| Pennsylvania, anthracite.....         | 57                | 1,121,603                         | ....                    | .....    | .....                 |
| Grand total (except Alabama).....     | 144               | 3,953,222                         | ....                    | .....    | .....                 |

(a) Only one steam shovel was in use in Alabama, and the extent of its operations cannot be revealed.

### *Steam-Shovel Operation*

Steam shovels operate either in a circular path beginning at the outside of the area and gradually working inward spirally until an area or island is left in the center about which the shovel cannot be economically moved (Fig. 14).<sup>4</sup>

This method is used mainly by the old-type fixed shovels to avoid turning the shovel, but with the modern revolving shovel this necessity

<sup>4</sup> The illustrations for the methods of operating steam shovels are furnished by Grant Holmes of Danville, Ill.



is obviated and such shovels make first a "thorough cut" in a straight line across the property (Fig. 15), dumping the spoil on the surface, and

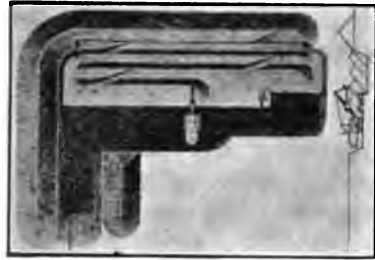


FIG. 17.—MODIFIED.



FIG. 16.—MODIFIED.



FIG. 15.—STRAIGHT.



FIG. 14.—SPIRAL.



FIG. 20.—CORNER OPENING.

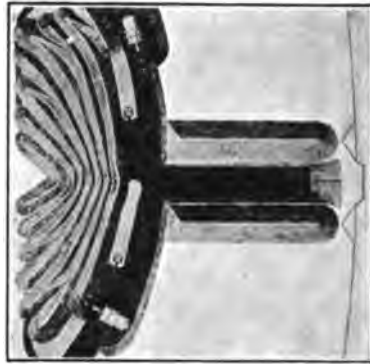


FIG. 19.—CORNER OPENING.

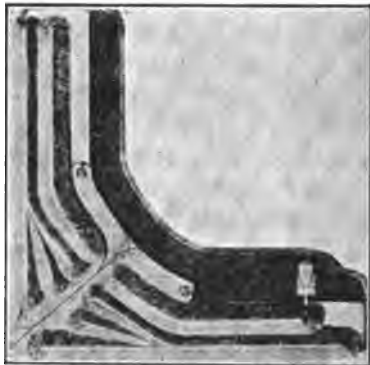


FIG. 18.—CORNER OPENING.

FIGS. 14 TO 20.—VARIOUS METHODS OF STEAM-SHOVEL OPERATION.

after reaching the boundary return on a parallel cut, dumping the spoil in the excavation from which the coal has been meanwhile removed, either by hand or by a smaller steam shovel.

The details of the parallel cut method can be modified as shown by Figs. 15 to 17, and as necessitated by the shape of the property, drainage, depth of overburden, etc.

Figs. 16 and 17 show two such modifications of the ordinary parallel system. The first cut is made close to one of the property lines and extends across the property. At the far end cuts are made at right angles extending to the other property line. When the shovel reaches the far property line it reverses its direction and returns parallel to the previous cut. The cross ridges shown in Figs. 15 to 17 are made when the shovel turns into the bank to begin the return cut. After the shovel has dug in the proper distance in beginning the cut, it digs back to uncover the triangular piece left in opening up the cut. In Fig. 16 the small loading shovel is shown following the digging shovel and there is a curve in the loading track laid on top of the coal. This curve must be continually moved forward as the loading shovel advances and is, therefore, a constant source of expense and trouble. In Fig. 17 the digging shovel follows the loading shovel; the loading track is continuous along the coal face and therefore does not require changing.

Fig. 18 shows a stripping field opened from the corner and having a diagonal haulway. The shovel starts from the upper left-hand corner and makes the first cut along one property line, depositing the spoil away from the boundary line. It then returns to the starting corner and on its next cut dumps the spoil from this cutting and about one-half of the waste bank from the thorough-cut against line so as to prevent caving along the line. Then returning again to the original starting corner, a second thorough-cut is made at right angles to the first, a gap being left between the ends of the spoil bank at the starting corner to provide for future haulage. In order to avoid right-angle turns with the shovel, wedge-shaped cuts are taken off from the starting corner as indicated and soon a continuous face is formed, giving easy turns for the shovel. With such an arrangement two small shovels can be used, both loading coal or, if there is an underlying fire clay, as sometimes occurs, the second shovel can load fire clay. If the underlying material is of value, a larger output of coal can be obtained by using two stripping shovels and two loading shovels, but in this case it would not be necessary to provide for the small wedge-shaped cuts, as neither of the shovels would have to then make a right angle turn.

Fig. 19 shows a somewhat similar method of opening up by running the thorough-cut through the center of the property and across its full width. At the far end of this thorough-cut the shovel cuts in opposite directions, but by first making a number of short cuts a continuous face is produced as shown.

Fig. 20 shows a method intended to accomplish the same result as that illustrated by Fig. 22. From one corner a thorough-cut is made

along one property line, the overburden being thrown inward and away from the property line. When the opposite boundary is reached the shovel makes alternate short and long cuts in order that it may be kept working along a line  $45^{\circ}$  to the boundary, dumping being toward the boundary line. Another thorough-cut is made along the boundary at right angles to the first and a similar procedure followed, thus giving a fan-shaped area and insuring that spoil banks are left against each property line to protect them from caving.

#### MINING AND LOADING THE COAL

The thin layer of dirt left on the coal by the shovel is removed with hand shovels and brooms and vertical holes are then drilled in the coal



FIG. 21—MARION MODEL 36 LOADING SHOVEL.

by hand augers or power drills at intervals of about 6 ft. and a few feet back from the face. The Jackhammer drill with a dull drill bit is used in the Danville field. The Clipper Blast Hole drill is also used for this purpose. Wet holes are dried out by firing one-fourth of a stick of dynamite in the hole. The coal is shot by using about a quart of black powder in each hole, and a keg of powder gives about 110 tons of loose coal in the Kansas field.

At small operations the coal is still shoveled into pit cars by hand, but small revolving steam shovels are rapidly replacing hand labor for loading wherever conditions permit their use.

openings pass under the spoil bank deposited in the cut out to the face of the coal.

Strip mines located above underground mines may be drained through bore-holes into the under mine.

### COSTS

C. E. Leshner in the U. S. Geological Survey Report for 1915 says:

"That coal can be mined with steam shovels more cheaply than by underground methods is indicated by the fact that the average daily output per employee in the stripped pit operations is twice as great as the average for the respective States. In other words the labor cost is about one-half as great."

Reference has already been made in the introduction to the fact that too often such items as repairs, amortization, delays and expense incident to floods are overlooked in calculating the cost per ton of coal produced by strip mining.

In *The Colliery Engineer* for March, 1913, Barry Scobee gives the cost of removing an average of 17 ft. of soil in the Kansas field as 5 to 6 c. per cubic yard of dirt removed, the overburden consisting of 6 ft. of soil, 6 ft. of shale or soapstone and thin blue shale to the coal.

The steam-shovel crew consists of:

- 1 craneman,
- 1 oiler,
- 1 fireman,
- 1 coal shoveler.

In the pit there are usually three men.

The crew of an electrically operated shovel consists of:

- 1 engineer,
- 1 oiler,

and the pit crew of two or three men, thus decreasing the labor expense for a shovel so operated. There is also no expense for coal or water which must be compensated for by the cost of power.

The Marion company says "Strip costs as given to us are not very reliable, but we should say that 40 to 80 c. per ton of coal mined would be a fair estimate varying with conditions."

The Bucyrus company gives the following estimates of costs for its different types of shovels:

## 175-B Steam Shovel.

|                      |            |            |                                       |
|----------------------|------------|------------|---------------------------------------|
| 1 engineer           | per month, | \$155.00   | Coal uncovered per day at 2000 cu.    |
| 1 craneman           | per month, | 100.00     | yd. stripping on coal 3'-0" thick,    |
| 1 fireman            | per month, | 75.00      | being a ton of coal under each square |
| 4 pitmen             | per month, | 260.00     | yard of surface.                      |
| 1 watchman           | per month, | 75.00      |                                       |
| 140 tons coal        | per month, | 140.00     | 15' deep × 85'-0" wide 400 tons       |
| Water                | per month, | 50.00      | 18' deep × 80'-0" wide 334 tons       |
| Oil, waste, pack'g.  | per month, | 30.00      | 21' deep × 80'-0" wide 297 tons       |
| Repairs and upkeep   | per month, | 200.00     | 24' deep × 80'-0" wide 250 tons       |
| Share of supt.       | per month, | 75.00      | 27' deep × 75'-0" wide 222 tons       |
| Incidental           | per month, | 150.00     | 30' deep × 70'-0" wide 199 tons       |
| Interest on \$28,000 | per month, | 140.00     | 33' deep × 60'-0" wide 182 tons       |
|                      |            | <hr/>      |                                       |
|                      |            | \$1,450.00 | 35' deep × 50'-0" wide 171 tons       |

At the rate of stripping 40,000 cu. yd. per month at an expense of \$1450 it makes the rate  $3\frac{6}{10}$  c. per cubic yard.

We submit herewith an estimate on the expense per ton of stripping, taking out the coal, hauling it to the tipple, screening it and delivering it on railroad cars, our 175-B shovel to handle 24 ft. in depth of overburden at the rate of 2000 cu. yd. per day of 9 hr., uncovering 250 tons of coal per day, or 5000 tons per month of coal 3 ft. thick as follows:

| 175-B Shovel              |                    | Expense per Month |            |
|---------------------------|--------------------|-------------------|------------|
| Steam shovel.....         | 5,000 tons, \$0.29 | per ton.....      | \$1,450.00 |
| Blasting.....             | 5,000 tons, 0.05   | per ton.....      | 250.00     |
| Loading.....              | 5,000 tons, 0.14   | per ton.....      | 700.00     |
| Hauling.....              | 5,000 tons, 0.07   | per ton.....      | 350.00     |
| Tipple and track.....     | 5,000 tons, 0.07   | per ton.....      | 350.00     |
| Superintendent.....       | 5,000 tons, 0.03   | per ton.....      | 150.00     |
| Repairs.....              | 5,000 tons, 0.06   | per ton.....      | 300.00     |
| Incidental.....           | 5,000 tons, 0.03   | per ton.....      | 150.00     |
| Interest on \$40,000..... | 5,000 tons, 0.04   | per ton.....      | 200.00     |
|                           |                    | <hr/>             |            |
| Total.....                | 1.78               | per ton.....      | \$3,900.00 |

The expense of operating the 225-B shovel per month in Kansas and the amount of coal it will uncover in different depths of stripping at the rate of 4000 cu. yd. of stripping per day or 80,000 cu. yd. per month are here given as follows:

| 225-B Steam Shovel              |            |            |                                       |
|---------------------------------|------------|------------|---------------------------------------|
| 1 engineer                      | per month, | \$155.00   | Coal uncovered per day at 4000 cu.    |
| 1 craneman                      | per month, | 100.00     | yd. stripping on coal 3'-0" thick     |
| 1 fireman                       | per month, | 78.00      | being a ton of coal under each square |
| 4 pitmen                        | per month, | 270.00     | yard of surface.                      |
| 1 oiler                         | per month, | 68.00      |                                       |
| 1 watchman                      | per month, | 75.00      | 15'-0" deep × 115'-0" wide 798 tons   |
| 150 tons coal                   | per month, | 150.00     | 18'-0" deep × 110'-0" wide 669 tons   |
| Water                           | per month, | 80.00      | 21'-0" deep × 106'-0" wide 574 tons   |
| Oil, waste, pack'g              | per month, | 50.00      | 24'-0" deep × 100'-0" wide 499 tons   |
| Repair and upkeep               | per month, | 290.00     | 27'-0" deep × 95'-0" wide 444 tons    |
| Share of supt.                  | per month, | 100.00     | 30'-0" deep × 90'-0" wide 400 tons    |
| Incidental                      | per month, | 189.00     | 33'-0" deep × 84'-0" wide 364 tons    |
| Interest on \$39,000 per month, |            | 195.00     | 36'-0" deep × 75'-0" wide 333 tons    |
|                                 |            | <hr/>      |                                       |
|                                 |            | \$1,800.00 | 39'-0" deep × 65'-0" wide 310 tons    |

At the rate of stripping 80,000 cu. yd. per month at an expense of \$1800 the cost would be  $2\frac{1}{4}$  c. per cubic yard.

We submit herewith an estimate on the expense per ton of stripping, loading the coal by hand into 2-ton tram cars, hauling it to the tippie, screening it and delivering it on railroad cars, our 225-B shovel to handle 24 ft. in depth of overburden at the rate of 4000 cu. yd. per day of 9 hr. uncovering say 500 tons of coal per day 3 ft. thick or 10,000 tons per month as follows:

| 225-B Steam Shovel    |                     | Expense per Month |            |
|-----------------------|---------------------|-------------------|------------|
| Steam shovel.....     | 10,000 tons, \$0.18 | per ton.....      | \$1,800.00 |
| Blasting.....         | 10,000 tons, 0.05   | per ton.....      | 500.00     |
| Loading.....          | 10,000 tons, 0.14½  | per ton.....      | 1,450.00   |
| Hauling.....          | 10,000 tons, 0.08   | per ton.....      | 800.00     |
| Tippie and track..... | 10,000 tons, 0.08   | per ton.....      | 800.00     |
| Superintendent.....   | 10,000 tons, 0.02   | per ton.....      | 200.00     |
| Repairs.....          | 10,000 tons, 0.04½  | per ton.....      | 450.00     |
| Incidental.....       | 10,000 tons, 0.03   | per ton.....      | 300.00     |
| Interest.....         | 10,000 tons, 0.03   | per ton.....      | 300.00     |
|                       |                     | <hr/>             |            |
| Total.....            | \$0.66              | per ton.....      | \$6,600.00 |

The expense of operating our 150-B shovel per month in coal stripping in Kansas and the amount of coal it will uncover in different depths of stripping at the rate of 1500 cu. yd. of stripping per day or 30,000 cu. yd. per month, are here given as follows:

| 150-B Shovel         |            |          |
|----------------------|------------|----------|
| 1 engineer           | per month, | \$155.00 |
| 1 craneman           | per month, | 100.00   |
| 1 fireman            | per month, | 75.00    |
| 3 pitmen             | per month, | 190.00   |
| 1 watchman           | per month, | 75.00    |
| 70 tons coal         | per month, | 70.00    |
| 140,000 gal. water   | per month, | 50.00    |
| Oil, waste, pack'g   | per month, | 30.00    |
| Repairs and upkeep   | per month, | 100.00   |
| Share of supt.       | per month, | 75.00    |
| Incidental expense   | per month, | 100.00   |
| Interest on \$20,000 | per month, | 100.00   |

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\$1,125.00

At the rate of stripping 30,000 cu. yd. per month at an expense of \$1,125 makes the rate at  $3\frac{3}{4}$  c. per cubic yard.

We submit herewith an estimate on the expense per ton of stripping, taking out the coal, hauling it to the tipple, screening it and delivering it on railroad cars, our 150-B revolving shovel to handle 21 ft. in depth of overburden at the rate of 1500 cu. yd. per day of 9 hr., uncovering 200 tons of coal per day or 4000 tons per month of coal 3 ft. thick as follows:

| 150-B Shovel              |                                   | Expenses per Month |            |
|---------------------------|-----------------------------------|--------------------|------------|
| Steam shovel.....         | 4,000 tons, \$0.28                | per ton.....       | \$1,125.00 |
| Blasting.....             | 4,000 tons, 0.05                  | per ton.....       | 200.00     |
| Loading.....              | 4,000 tons, 0.14                  | per ton.....       | 560.00     |
| Hauling.....              | 4,000 tons, 0.07                  | per ton.....       | 280.00     |
| Tipple and track.....     | 4,000 tons, 0.07                  | per ton.....       | 280.00     |
| Superintendent.....       | 4,000 tons, 0.03 $\frac{3}{4}$    | per ton.....       | 150.00     |
| Repairs.....              | 4,000 tons, 0.04                  | per ton.....       | 160.00     |
| Incidental.....           | 4,000 tons, 0.03 $\frac{1}{2}$    | per ton.....       | 140.00     |
| Interest on \$16,000..... | 4,000 tons, 0.02                  | per ton.....       | 80.00      |
| Total.....                | \$0.74 $\frac{1}{4}$ per ton..... |                    | \$2,970.00 |

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## A Study of the Microstructure of Some Clays in Relation to Their Period of Firing

BY H. RIES\* AND Y. OINOUE†

(St. Louis Meeting, October, 1917)

### INTRODUCTION

Of the several interesting physical properties of clay which have claimed the attention of investigators in recent years, none is more important than the behavior of the material when heated to temperatures above those of dehydration.

Some of the problems to be solved are:

1. The mineralogical changes that take place in firing.
2. The relation between temperature and change in constitution, shrinkage, porosity, etc.
3. The importance of the time factor, as duration of firing, either in reaching a given temperature, or of exposure to a constant temperature.

### *Mineralogical Changes*

As a result of fusion, reactions take place between the constituents of a clay, which results in a reorganization of the mass, and the formation of new compounds. Of the new minerals developed, sillimanite is the one usually recorded, and has been referred to by a number of observers.<sup>1</sup> There appears to be a difference of opinion, however, regarding the temperature at which it is formed. Thus Klein<sup>2</sup> notes that sillimanite forms only above 1300° C., while Mellor claims that it can develop below 1200° C., the difference in results being conditioned by the time factor. In Klein's experiments the test pieces were fired in 12 hr., while in Mellor's, the temperature rose from 800° to 1180° in 24 hrs., and occupied the same time in cooling back to 800°.

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<sup>1</sup> Vernadsky: *Bulletin de la Société Française de Minéralogie* (1890), 13, 256; Mellor: *Journal of the Society of Chemical Industry* (1907), 26, 375; Klein: *U. S. Bureau of Standards, Technologic Paper No. 80* (1916), 36; and others.

<sup>2</sup> *Transactions of the English Ceramic Society* (1916-17), 16.

Mellor and Holdcroft<sup>3</sup> state that when clays are fired at 600°, the molecule probably breaks down in free alumina, and free silica, while ferruginous clays also form  $\text{Fe}_2\text{O}_3$ . Hussak<sup>4</sup> found that low-burned porcelains show all the constituents doubly refracting and clearly distinguishable from each other, but that higher heating gave an amorphous groundmass, with a few large quartz grains, and needles that were apparently sillimanite.

Klein found that by heating English kaolin for 5 hr. at 600° C., the kaolinite became entirely isotropic, but that the index of refraction (1.55) was only slightly altered. Further, he says that no profound change was noted in the optical properties until 1200° C. was reached, at which point there began incipient dissociation of the kaolinite, into two isotropic substances, with complete dissociation at 1400° C, and the development of prismatic grains between 1400° C. and 1450° C.

There is thus a difference of 600° C. in the dissociation point of kaolinite as expressed by Mellor and Klein and there may also be some question as to whether the change from an anisotropic to an isotropic condition does not indicate a change in the mineral constitution.

Another possible change referred to in recent papers is that of quartz to tridymite and cristobalite, but on this point the results obtained by different observers vary, and may be due to the influence of the time factor. McDowell<sup>5</sup> found that upon repeatedly burning silica brick the quartz is transformed to tridymite and cristobalite.

Where we are dealing with impure clays, changes, due to fluxing action will undoubtedly begin at a much lower temperature than they do in the purer clays like kaolin.

### *Temperature and Time Factors*

It is commonly recognized that as the temperature increases, shrinkage and density increase up to a certain point, beyond which they diminish, but that the time factor is an important one. In other words, certain changes produced by heating a clay to a given temperature in a given time, may be reached by taking a longer time to heat the same clay to a lower temperature or by holding it at the latter for a considerable period. This is thought to explain the difference of view regarding the formation of sillimanite expressed by Mellor and Klein, although the latter says that repeated burning of a clay at the same temperature does not produce much change in its constitution.

A criticism sometimes made of laboratory tests on clay, is that they lack the long exposure to a slowly rising temperature, obtained in a

<sup>3</sup> *Transactions of the English Ceramic Society* (1911), 9, 91.

<sup>4</sup> *Sprechsaal* (1889), 153.

<sup>5</sup> *Bulletin* No. 119 (November, 1916), 1999.

regular factory kiln. While this is true, it is also true that the test pieces used in the laboratory may be smaller than those fired in a large kiln, and require less time to heat through.

The tests in the present paper contain some data on this subject.

### SCOPE OF PRESENT PAPER

Most of the petrographic work done on burned clays has been carried out on high-grade clays which contain relatively small amounts of fluxing impurities, and, moreover, the studies have been confined chiefly to results obtained at temperatures of 1200° C. or over.

The present paper deals with some clays that contain notable amounts of fusible impurities,<sup>6</sup> and were fired at temperatures not exceeding 1150° C.

### METHOD OF PROCEDURE

In the present investigation, which is to be regarded as a preliminary one, the clays tested were molded in 1-in. (25.4-mm.) cubes, and 3-in. (76.2-mm.) bars. One set was heated to 1000° C. in 1½ hr. (referred to on subsequent pages as the preheating period), and held there for 10 hr.; a second set was heated to 1150° C. in 3 hr., and held at that temperature for 10 hr.

Samples were drawn at the end of the preheating period, and at hourly intervals thereafter, the test pieces after removal from the furnace being buried in hot sand, so as to cause slow cooling.

The pieces so fired were then tested for fire shrinkage, absorption, porosity. Thin sections were also cut for petrographic study.

The tensile strength of both the air-dried and burned clays was also tested.

### *Clays Selected*

The clays selected for investigation were the following:

1. Brown residual clay from Ordovician shale, occurring at Webster, Botetourt County, Va.

2. Grayish-white residual clay, derived presumably from Cambrian shale, occurring near Lofton, Augusta County, Va.

*Brown Clay.*—The Webster clay is soft, smooth, yellowish-brown when wet, and contains small fragments of unweathered shale. It grinds easily, and mixes with water to a fairly plastic mass. Under the microscope it shows minute flakes of hydromica<sup>7</sup> and kaolinite, as well as small particles of iron oxide. Scattered angular fragments of quartz were

<sup>6</sup> The microstructure of some impure shales after burning has been described by Teetor, *Transactions of the American Ceramic Society* (1916), 18, 843, but these were fired at Cone 3 (1190° C.).

<sup>7</sup> See Galpin: *Transactions of the American Ceramic Society*, 14, 306-307.

also noticed. Biotite is rare. The hydromica and kaolinite in the shale fragments were arranged parallel to the bedding planes of the material.

*White Clay.*—This is very fine-grained, smooth, has a greasy feel, and high plasticity. The constituents seen under the microscope were light brown aggregates of very minute kaolinite scales, angular fragments of colorless quartz, together with small amounts of red iron oxide and occasional grains of brown biotite, yellow epidote, zircon, zoisite, and tourmaline.

### *Properties of the Clays*

The chemical composition of the two clays is shown by the following analyses:<sup>8</sup>

|   | Brown    | White        |
|---|----------|--------------|
| SiO <sub>2</sub> .....                              | 51.08    | 52.10        |
| Al <sub>2</sub> O <sub>3</sub> .....                | 22.05    | 31.54        |
| Fe <sub>2</sub> O <sub>3</sub> .....                | 10.43    | 2.38         |
| CaO.....  | .....    | 0.10         |
| MgO.....  | 2.28     | 0.58         |
| Na <sub>2</sub> O.....                              | 0.20     | 0.27         |
| K <sub>2</sub> O.....                               | 5.92     | 1.89         |
| H <sub>2</sub> O.....                               | 7.36     | 10.89        |
|   | <hr/>    | <hr/>        |
| Total fluxes.....                                   | 99.32    | 99.75        |
|   | <hr/>    | <hr/>        |
|   | Per cent | Per cent.    |
| Amount of water required to develop plasticity..... | 44.0     | 47.7         |
| Plasticity.....                                     | Fair     | High         |
| Air shrinkage.....                                  | 4.5      | 8.1          |
| Color after firing.....                             | Red      | Creamy white |

### *Effects of Firing*

*Color.*—The brown clay shows a gradual oxidation of the iron oxide, the color becoming darker and darker, though not strongly so, as the time of burning progressed.

The white clay shows no such pronounced change of color, owing to the small amount of iron oxide which it contains.

*Hardness.*—The longer the heating, the more compact and harder the clays became, due probably to progressive fluxing of kaolinite and hydromica with the cementing material, and with this there appeared to be a decrease in size of the pore spaces.

*Loss on Ignition.*—The driving off of chemically combined water or other volatile substances seems to have been practically completed, as was to be expected, during the stage of rising temperature (preheating

<sup>8</sup> We are indebted to Dr. T. L. Watson, State Geologist of Virginia, for permission to use these analyses in advance of their publication in a bulletin of the Virginia Geological Survey.

period) and so the amount of loss on ignition of the different samples drawn from the fire during the periods of constant temperature remains practically the same (Figs. 1 to 4).

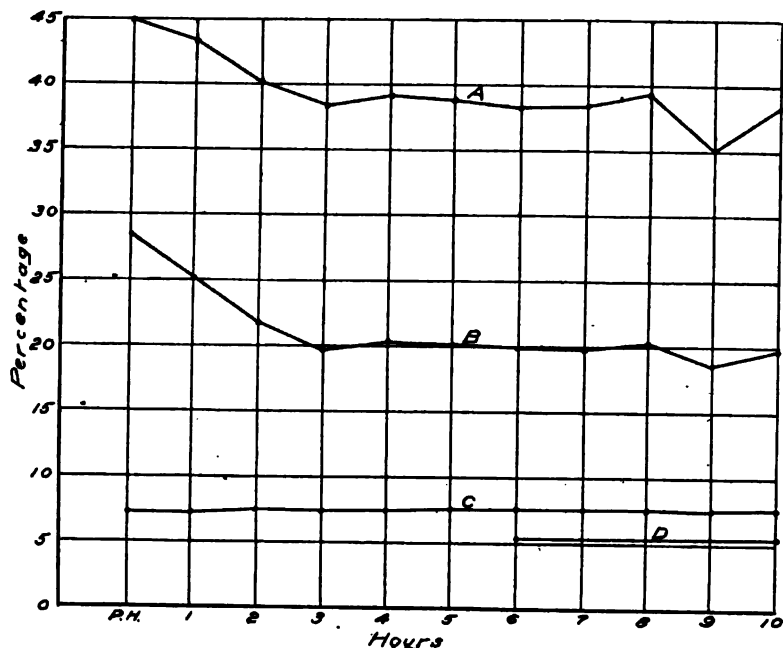


FIG. 1.—BROWN CLAY FIRED AT 1000° C. A, POROSITY; B, ABSORPTION; C, LOSS OF WEIGHT ON IGNITION; D, FIRE SHRINKAGE.

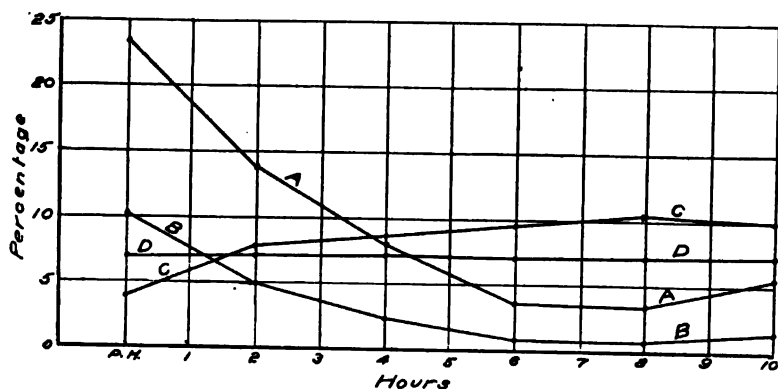


FIG. 2.—BROWN CLAY FIRED AT 1150° C. LETTERS ON CURVES SAME AS FIG. 1.

*Absorption and Porosity.*—The absorption was determined by soaking for 48 hr., while the porosity was found after a further boiling for 3 hrs., this giving as good results as immersion in a vacuum. It will be seen by reference to Figs. 1 to 4, that the absorption and porosity curves run practically parallel with each other, the decreasing amount being a

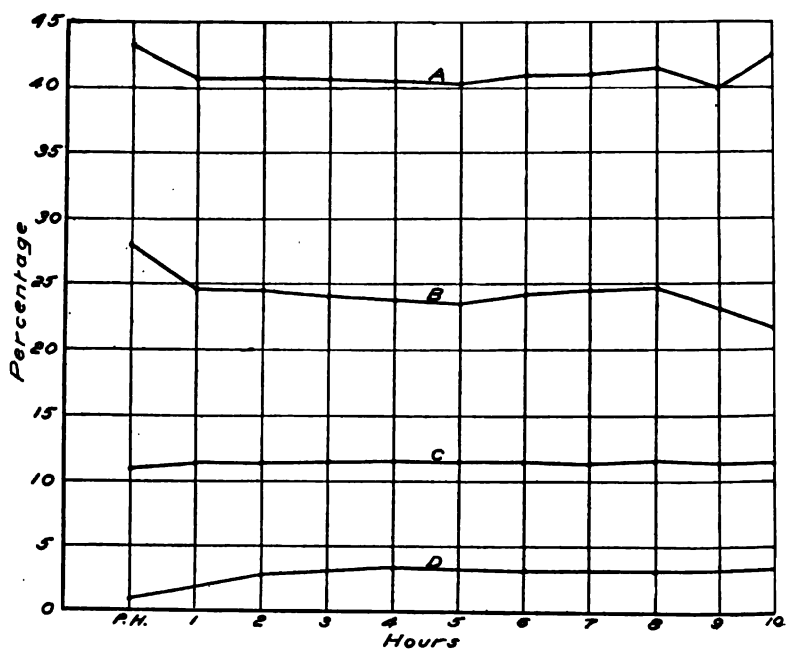


FIG. 3.—WHITE CLAY FIRED AT 1000° C. LETTERS ON CURVES SAME AS FIG. 1.

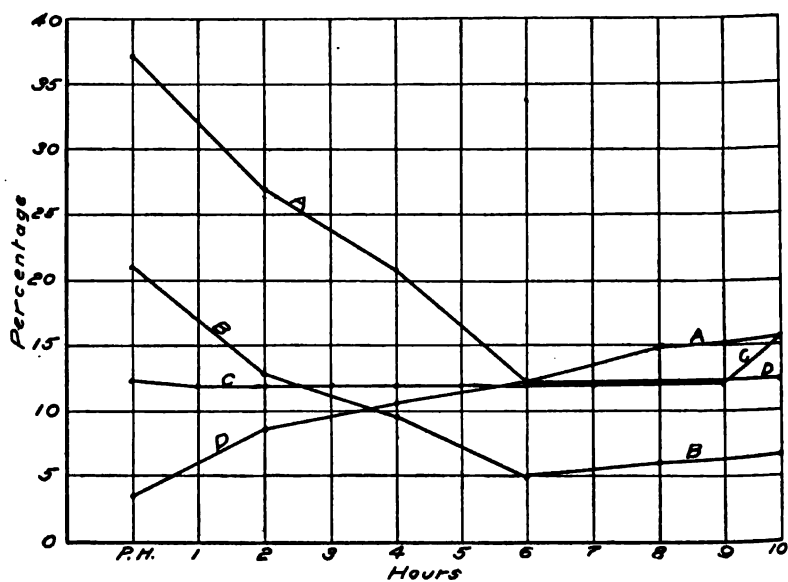


FIG. 4.—WHITE CLAY FIRED AT 1150° C. LETTERS ON CURVES SAME AS FIG. 1.



factor of temperature due to the shrinkage of the pore space in the clays, and fusion.

In the brown clay, both absorption and porosity decreased rapidly until the end of 3 hr., and after that they fell off very slowly.

The white clay behaved somewhat differently from the brown, there being no great change at 1000° C. but a rapid decrease in absorption and porosity at 1150° C. which decreased with the time of burning until near the end when it seemed to rise slightly.

As a matter of fact, both the absorption and the porosity decreased to a certain limit and then began to increase slightly in the tenth hour.

It is a little difficult to explain this increase in the final hour, for this clay does not appear to have reached its vitrification point (Figs. 1 to 4).

Tables showing properties of the burned clays.

TABLE 1.—*Brown Clay Fired at 1000° C.*

|                           | Color                  | Hard-<br>ness | Loss on Igni-<br>tion,<br>Per Cent. | Absorp-<br>tion,<br>Per Cent. | Poros-<br>ity,<br>Per Cent. | Fire<br>Shrinkage,<br>Per Cent. |
|---------------------------|------------------------|---------------|-------------------------------------|-------------------------------|-----------------------------|---------------------------------|
| At the end of preheating  | yellowish<br>brick red | 2.5           | 7.17                                | 27.89                         | 44.95                       |                                 |
| At the end of 1 hr. ....  | ↑                      | 3.0           | 7.14                                | 25.18                         | 43.25                       |                                 |
| At the end of 2 hr. ....  |                        | ↑             | 7.47                                | 21.77                         | 40.09                       |                                 |
| At the end of 3 hr. ....  |                        |               | 7.40                                | 19.60                         | 37.92                       |                                 |
| At the end of 4 hr. ....  |                        |               | 7.43                                | 20.53                         | 39.16                       |                                 |
| At the end of 5 hr. ....  | darker                 | harder        | 7.58                                | 20.34                         | 38.86                       |                                 |
| At the end of 6 hr. ....  | ↓                      |               | 7.64                                | 19.87                         | 38.31                       | 5.34                            |
| At the end of 7 hr. ....  |                        |               | 7.60                                | 19.78                         | 38.39                       |                                 |
| At the end of 8 hr. ....  |                        |               | 7.60                                | 20.57                         | 39.37                       |                                 |
| At the end of 9 hr. ....  | ↓                      | ↓             | 7.49                                | 18.63                         | 35.09                       |                                 |
| At the end of 10 hr. .... | brick red              | 5.0           | 7.63                                | 19.80                         | 38.25                       | 5.34                            |

TABLE 2.—*Brown Clay Fired at 1150° C.*

|                           | Color         | Hard-<br>ness        | Loss on Igni-<br>tion,<br>Per Cent. | Absorp-<br>tion,<br>Per Cent. | Poros-<br>ity,<br>Per Cent. | Fire<br>Shrinkage,<br>Per Cent. |
|---------------------------|---------------|----------------------|-------------------------------------|-------------------------------|-----------------------------|---------------------------------|
| At the end of preheating  | light red     | steel hard           | 6.78                                | 10.18                         | 23.40                       | 3.9                             |
| At the end of 1 hr. ....  | ↑             | ↑                    | 7.06                                |                               |                             |                                 |
| At the end of 2 hr. ....  |               |                      | 7.01                                | 4.93                          | 13.81                       | 7.8                             |
| At the end of 3 hr. ....  |               |                      | 7.11                                |                               |                             |                                 |
| At the end of 4 hr. ....  | darker        | harder               | 7.15                                | 2.26                          | 7.90                        | 8.7                             |
| At the end of 5 hr. ....  | ↓             | ↓                    | 7.16                                |                               |                             |                                 |
| At the end of 6 hr. ....  |               |                      | 7.13                                | 0.78                          | 3.65                        | 9.6                             |
| At the end of 7 hr. ....  |               |                      | 7.17                                |                               |                             |                                 |
| At the end of 8 hr. ....  |               |                      | 7.24                                | 0.75                          | 3.53                        | 10.0                            |
| At the end of 9 hr. ....  | ↓             | ↓                    | 7.22                                |                               |                             |                                 |
| At the end of 10 hr. .... | dark<br>brown | harder<br>than steel | 7.28                                | 1.49                          | 5.53                        | 9.9                             |

TABLE 3.—*White Clay Fired at 1000° C.*

|                          | Color       | Hardness | Loss on Ignition, Per Cent. | Absorption, Per Cent. | Porosity, Per Cent. | Fire Shrinkage, Per Cent. |
|--------------------------|-------------|----------|-----------------------------|-----------------------|---------------------|---------------------------|
| At the end of preheating | slight pink | 3.5      | 10.86                       | 27.95                 | 43.19               | 1.00                      |
| At the end of 1 hr.....  | "           | 4.0      | 11.37                       | 24.62                 | 40.67               |                           |
| At the end of 2 hr.....  | "           | ↑        | 11.40                       | 24.53                 | 40.81               | 2.83                      |
| At the end of 3 hr.....  | "           |          | 11.53                       | 24.14                 | 40.72               |                           |
| At the end of 4 hr.....  | "           |          | 11.58                       | 23.82                 | 40.46               | 3.47                      |
| At the end of 5 hr.....  | "           | harder   | 11.50                       | 23.47                 | 40.32               |                           |
| At the end of 6 hr.....  | "           |          | 11.53                       | 24.16                 | 40.87               | 3.17                      |
| At the end of 7 hr.....  | "           |          | 11.37                       | 24.58                 | 40.95               |                           |
| At the end of 8 hr.....  | "           |          | 11.58                       | 24.65                 | 41.49               | 3.17                      |
| At the end of 9 hr.....  | "           |          | 11.42                       | 23.21                 | 39.92               |                           |
| At the end of 10 hr..... | "           | 5.0      | 11.36                       | 21.70                 | 42.49               | 3.37                      |

TABLE 4.—*White Clay Fired at 1150° C.*

|                          | Color                | Hardness          | Loss on Ignition, Per Cent. | Absorption, Per Cent. | Porosity, Per Cent. | Fire Shrinkage, Per Cent. |
|--------------------------|----------------------|-------------------|-----------------------------|-----------------------|---------------------|---------------------------|
| At the end of preheating | slight pink          | 4.5               | 12.33                       | 21.10                 | 37.18               | 3.6                       |
| At the end of 1 hr.....  | "                    | harder than steel | 11.90                       |                       |                     |                           |
| At the end of 2 hr.....  | "                    | ↑                 | 11.85                       | 12.76                 | 26.75               | 8.5                       |
| At the end of 3 hr.....  | "                    |                   | 11.93                       |                       |                     |                           |
| At the end of 4 hr.....  | "                    |                   | 11.93                       | 9.63                  | 21.00               | 10.6                      |
| At the end of 5 hr.....  | pinkish creamy white | harder            | 12.00                       |                       |                     |                           |
| At the end of 6 hr.....  | "                    |                   | 12.01                       | 4.95                  | 12.31               | 12.3                      |
| At the end of 7 hr.....  | "                    |                   | 12.01                       |                       |                     |                           |
| At the end of 8 hr.....  | "                    |                   | 12.09                       | 6.06                  | 14.94               | 12.3                      |
| At the end of 9 hr.....  | "                    |                   | 12.21                       |                       |                     |                           |
| At the end of 10 hr..... | "                    | harder than steel | 15.54                       | 6.49                  | 15.56               | 12.5                      |

*Fire Shrinkage.*—The figures given in the tables show that in the series heated to 1000° C., the shrinkage remains practically constant after 3 hrs. at that temperature, but that in the 1150° C. series the maximum shrinkage takes a little longer to develop.

Moreover, the fire shrinkage may show little change, even though there is a visible change in porosity, absorption, and hardness (Figs. 1 to 4).

*Tensile Strength.*—The tensile strength of the burned clays shows some interesting changes depending upon the time of burning.

The brown clay (Fig. 5) showed a tensile strength of 370 lb. per square inch after 5 hrs. at  $1000^{\circ}\text{C}$ . and 430 lb. at the end of 10 hrs. In the case

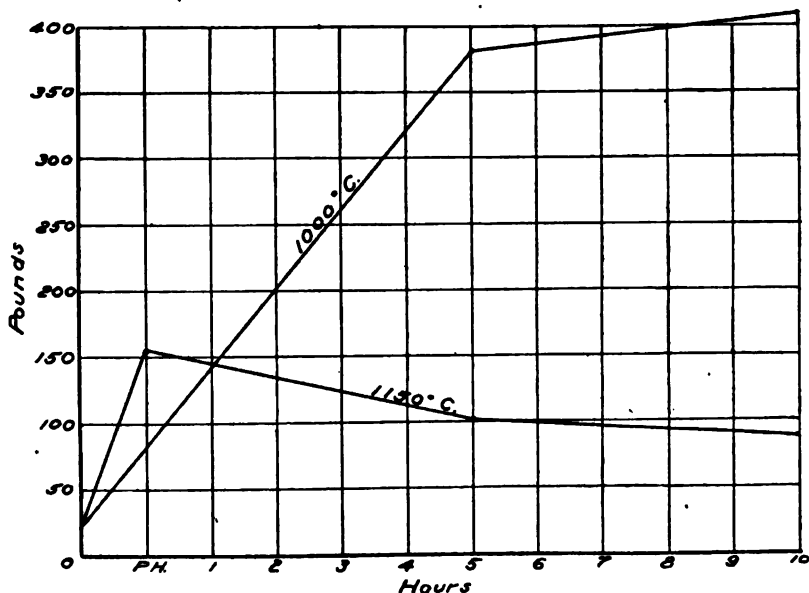


FIG. 5.—TENSILE STRENGTH OF BROWN CLAY AIR DRIED, AND AFTER FIRING.

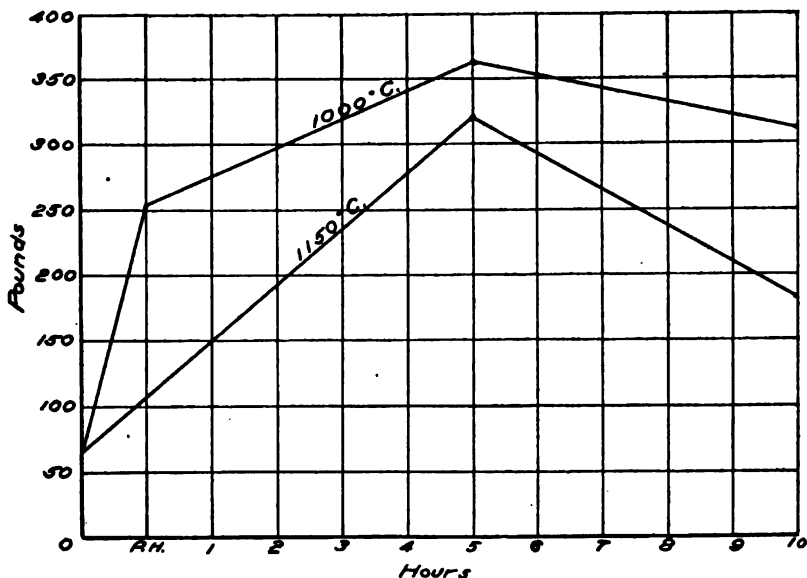


FIG. 6.—TENSILE STRENGTH OF WHITE CLAY AIR DRIED, AND AFTER FIRING.

of the briquettes heated to  $1150^{\circ}$ , the tensile strength was 153 lb. per square inch when this temperature was first reached and then fell off to 80 lb. at the end of 10 hrs.

The white clay shows a maximum tensile strength at the end of 5 hrs. heating at the same temperatures. The specimen heated to 1000° C. gave 360 lb. per square inch, while the one heated to 1150° C. showed 330 lb. per square inch.

### *Microscopic Examination of the Burned Clays*

In order to study the pyrometric changes, a number of thin sections of the burned cubes were examined under the microscope.

*Brown Clay.*—Under the microscope this reveals what at first appeared to be a porphyritic texture (Figs. 7 and 8), but it was found that the phenocryst-like grains were small fragments of shale that had escaped pulverizing, while the groundmass is a brownish mixture of iron oxide with fine particles of kaolinite and hydromica.

These shale fragments are no doubt portions of the original shale, not completely decomposed, and hence they contain less free iron oxide than the other parts of the mass. They are much lighter in color than the groundmass and consist of minute grains of quartz, and small flakes of kaolinite and hydromica, the last two showing a distinct parallel arrangement, which is quite clear when viewed under the microscope, but does not come out very distinctly in the photographs (Fig. 7). The hydromica shows a higher interference color than the normal one and may be a sericite.

Small amounts of quartz are scattered through the groundmass.

At 1000° C. the specimens indicate no appreciable change in texture, but there seems to be a slight decrease of the scaly minerals and an increase in the amount of amorphous (isotropic) material toward the end of the burning.

The color as a whole became darker and the material seemed to be more tightly cemented in the specimens removed at the end of the firing period.

Vitrification seems to have begun at the end of 3 hrs. exposure to 1150° C. and at this point the color changed to brownish-red and still later to reddish-brown.

The minute flakes of kaolinite and hydromica also seem to be disappearing, due in part to the fluxing of these minerals with the finer-grained ingredients, in which change the hydromica seems to flux more rapidly than the kaolinite. Their disappearance may also be due in part to their becoming isotropic under rising temperature. If so, the change was not complete at 1000° C.

In the shale fragments referred to above, there was noticeably less anisotropic material after 1 hr. at 1150° than there was after 1 hr. at 1000° C. Corrosion of the quartz fragments was also noticed at 1150° C., but it less was noticeable than in the case of the white clay.

When the thin sections are examined with polarized light, it can be

seen in passing from the specimens heated for 1 hr. at  $1000^{\circ}$  to those heated for 10 hrs. at  $1150^{\circ}$  that there is a gradual increase in the quantity

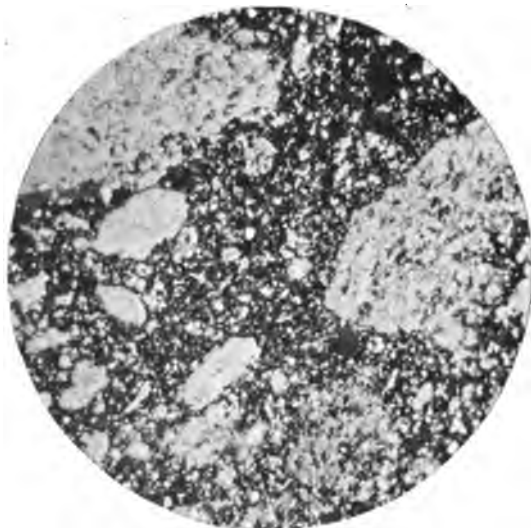


FIG. 7.—PHOTOMICROGRAPH OF THIN SECTION OF BROWN CLAY, AFTER HEATING 1 HR. AT  $1000^{\circ}$  C. LARGE GRAINS, PIECES OF PARTIALLY WEATHERED UNGROUND SHALE. BLACK MATERIAL MOSTLY ISOTROPIC.  $\times 50$ , WITH CROSSED NICOLS.

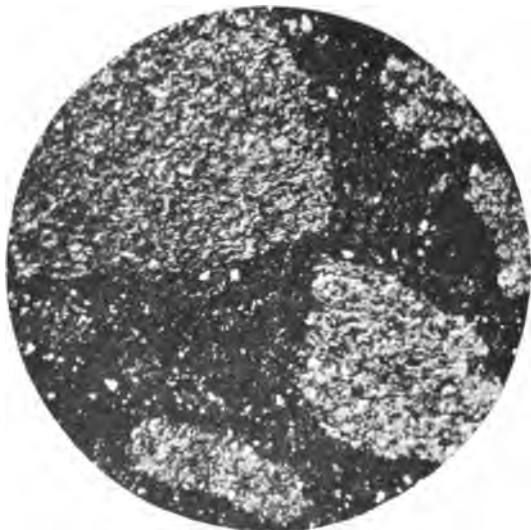


FIG. 8.—SAME CLAY AS FIG. 7, BUT HEATED FOR 10 HR. AT  $1150^{\circ}$  C. NOTE GREATER AMOUNT OF ISOTROPIC MATERIAL AS COMPARED WITH PRECEDING.  $\times 50$ , CROSSED NICOLS.

of isotropic material. This is somewhat difficult to show in a photograph, as the halation produced by the grains still yielding light colors tends to

give the whole field a lighter tone. However, comparison of Figs. 7 and 8 will serve to show the contrast.

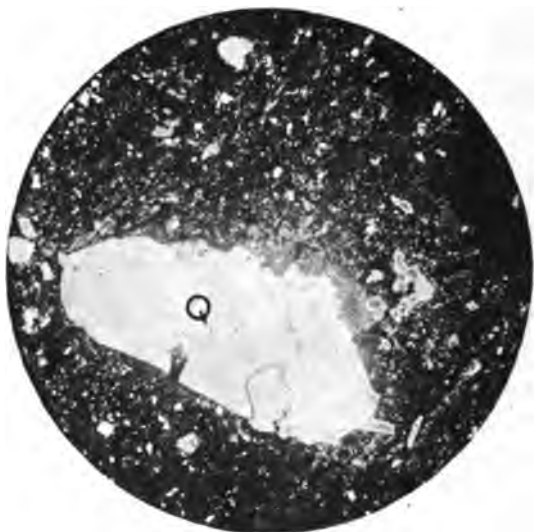


FIG. 9.—PHOTOMICROGRAPH OF THIN SECTION OF WHITE CLAY, AFTER HEATING 10 HR. AT 1000° C. Q, QUARTZ, SOMEWHAT CORRODED BY FUSION. DARK GROUND MOSTLY ISOTROPIC.  $\times 50$ , WITH CROSSED NICOLS.

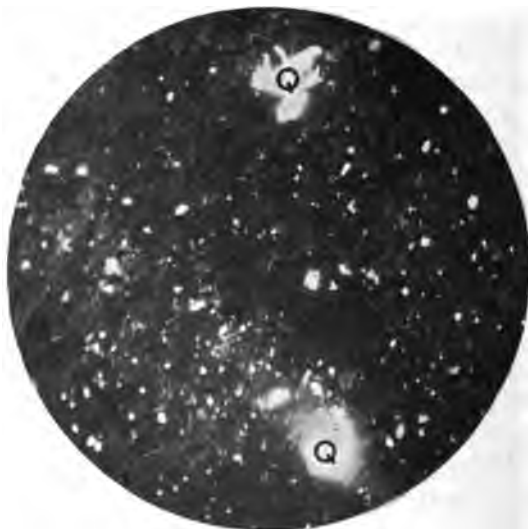


FIG. 10.—SAME CLAY AS FIG. 9, HEATED 10 HR. AT 1150° C. Q, CORRODED QUARTZ  $\times 50$ , CROSSED NICOLS.

As the fusion of hydromica and kaolinite proceeds, it results apparently in the formation of an iron silicate; volume changes also seem to occur resulting in the development of pore space in the clays.

*White Clay.*—Thin sections of the fired material showed a somewhat banded texture which is not an original structure of the clay, but due to the pressure used in molding; they also showed small fragments of quartz scattered through the very fine-grained aggregates of kaolinite and hydromica.

Small grains of iron oxide were also noticed as well as tiny fragments of biotite, epidote, zoisite, zircon and tourmaline.

At 1000° C. the minute flakes of kaolinite and hydromica seem to be altered to a light brown amorphous body and the disappearance of these minerals seems more noticeable as the exposure to the temperature above mentioned continues.

Toward the end of the heating at 1000° C. and 1150° C., corrosive action of the quartz was quite remarkable (Figs. 9 and 10) and the change in color to a darker hue was also evident. At 1150° C., the gradual disappearance of the kaolinite and hydromica after heating 10 hrs. at this temperature was very manifest. The edges of the quartz were more deeply eaten away according to the duration of burning as shown in the figures.

TABLE 5  
BROWN CLAY

|                  | 1000° C.                                     |                      |                      |                   | 1150° C.                                     |                      |                      |                   |
|------------------|--|----------------------|----------------------|-------------------|--|----------------------|----------------------|-------------------|
|                  | Iron Oxide and Isotropic Material, Per Cent. | Hydromica, Per Cent. | Kaolinite, Per Cent. | Quartz, Per Cent. | Iron Oxide and Isotropic Material, Per Cent. | Hydromica, Per Cent. | Kaolinite, Per Cent. | Quartz, Per Cent. |
| Raw clay.....    | 40   | 28                   | 27                   | 5                 |  |                      |                      |                   |
| 1 hr. burning.   | 50   | 20                   | 15                   | 5                 | 65   | 3.0                  | 17.0                 | 5                 |
| 5 hrs. burning.  | 55   | 15                   | 15                   | 5                 | 76   | 1.0                  | 5.0                  | 5                 |
| 10 hrs. burning. | 60   | 12                   | 10                   | 5                 | 79   | 0.5                  | 5.5                  | 3                 |

WHITE CLAY

|                  | 1000° C.             |                   |                      |                       |                               | 1150° C.             |                   |                      |                       |                               |
|------------------|----------------------|-------------------|----------------------|-----------------------|-------------------------------|----------------------|-------------------|----------------------|-----------------------|-------------------------------|
|                  | Kaolinite, Per Cent. | Quartz, Per Cent. | Hydromica, Per Cent. | Iron Oxide, Per Cent. | Isotropic Material, Per Cent. | Kaolinite, Per Cent. | Quartz, Per Cent. | Hydromica, Per Cent. | Iron Oxide, Per Cent. | Isotropic Material, Per Cent. |
| Raw clay..       | 63                   | 25                | 10                   | 2                     |                               |                      |                   |                      |                       |                               |
| 1 hr. burning.   | 53                   | 25                | 10                   | 2                     | 10                            | 19                   | 25                | 1.0                  | 2                     | 50.0                          |
| 5 hrs. burning.  | 35                   | 25                | 8                    | 2                     | 30                            | 9                    | 25                | 1.0                  | 2                     | 60.0                          |
| 10 hrs. burning. | 35                   | 25                | 8                    | 2                     | 30                            | 9                    | 25                | 0.5                  | 2                     | 60.5                          |

In one sample which had been heated up to  $1310^{\circ}\text{C}$ ., the corrosion of the quartz was very prominent, and at this point the clay seemed to show slight vitrification. However, no optical change was noticed in quartz up to this temperature.

In order to obtain some idea of the mineral content of the raw clays and the change in amount of each after heating, an attempt was made to estimate the per cent. of this material present. The figures given in table 5 are a rough quantitative estimate made from a study of the thin sections.

### SUMMARY

From the foregoing data, we see that when fusion begins in the mass the fine mineral grains are first affected and seem to change from a crystalline to an amorphous condition.

There is a gradual change in the color as the time of heating at one temperature continues.

The hardness gradually increases on longer burning up to steel hardness.

Absorption and porosity in each case run practically parallel, decreasing to a certain limit and then increasing again slightly, later, in the case of the clays studied. Absorption and porosity abruptly decrease when the mass approaches vitrification, as shown in the case of the white clay at  $1150^{\circ}\text{C}$ .

Loss on ignition is the same at both temperatures, as might be expected.

The fire shrinkage in both cases shows no change in the last 5 hrs. of continuous heating, while in the first 5 hrs. of heating at one temperature it shows a marked increase.

The tensile strength seems to reach a maximum point after 5 hrs. of continuous heating and decreases beyond that point.

Microscopic study shows that the longer the burning the denser the texture up to a certain point and that this is also accompanied in the brown clay by a change in color. There is, furthermore, in both clays a gradual increase in the amount of isotropic material.

Fusion of the hydromica and kaolinite proceed parallel with each other, but the former is more readily fusible.

Corrosive action as seen in the quartz increases with the length of burning.

Toward the end of the heating at  $1150^{\circ}\text{C}$ ., pore spaces begin to develop, especially in the case of the brown clay.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## A Review of the Exploration at Belle Isle, Louisiana

BY A. F. LUCAS,\* WASHINGTON, D. C.

(St. Louis Meeting, October, 1917)

### Introduction

BELLE ISLE, located in the low sea marshes near Atchafalaya Bay, is the southeasternmost of the famous Five Salt Islands of Louisiana. Rising about 80 ft. (24 m.) above the level of the surrounding marsh and being some 800 acres (323 ha.) in extent, it constitutes a striking landmark on the low Gulf coast, and since the eighteenth century, when it played an important part in the operation of the buccaneers, it has been intermittently frequented by man. In 1897, the writer discovered the great mass of salt that underlies the island and shortly afterward published a brief account of the geology of Belle Isle and the neighboring salt islands.<sup>1</sup> In that paper he described principally the salt, but pointed out also the occurrence of petroleum and of sulphur in the strata above the salt. Since that time the island has been extensively explored by the drill, and the developments up to 1908 have been described by Veatch<sup>2</sup> and Harris,<sup>3</sup> and Knapp.<sup>4</sup>

The writer's early conclusions as to the probable presence of sulphur in considerable quantity have been confirmed by the later drilling, some of the wells having found quantities of gas containing sulphur dioxide and a number of sulphur-bearing strata as well. Under the stimulus of the present keen demand for sulphur for war purposes, a syndicate of New York capitalists was recently formed to explore the sulphur resources of the island in more detail. Under the writer's direction, six wells were drilled for this syndicate during the winter of 1916-17. The present paper presents briefly the results of this work and sums up the present knowledge of the geology of this highly interesting locality.

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\* Consulting Engineer.

<sup>1</sup> A. F. Lucas: Rock Salt in Louisiana. *Trans.* (1899), 29, 462-474.

<sup>2</sup> A. C. Veatch: *Louisiana Geological Survey Report* (1899), 221-229.

<sup>3</sup> G. D. Harris: *Louisiana Geological Survey, Bulletin* No. 7 (1907), 18-26; *U. S. Geological Survey, Bulletin* No. 429 (1910), 43-48.

<sup>4</sup> I. N. Knapp: *Journal of the Franklin Institute* (November and December, 1912), 174, 447, 639.

*History of Development*

The first well drilled was near the old landing, in November, 1896. It found no salt at 590 ft. (179.8 m.) (Lucas 1), but the second well, (Lucas 2), located in the center of the northern section of the island, reached salt at the depth of 335 ft. (102 m.). No. 3, now in the canal, and No. 4 reached salt at 135 and 325 ft. (41 and 99 m.), respectively. No. 3 found oil of a canary yellow color 37° B<sub>é</sub>. at a depth of about 125 ft. (38 m.), and No. 4, south of it, found some oil with considerable gas at 335 ft. (102 m.), where it stopped in salt.

The Gulf company shortly afterward put down 13 wells to determine the contour of the salt surface, and in August, 1898, started a shaft close to the site of their prospect well No. 11, which found incipient salt at 103 ft. (31.4 m.) and solid salt at 140 ft. (42.6 m.). A shaft was carried to a depth of about 390 ft. (118.8 m.), at which level an entry was driven in the salt to the southwest and then to the south-southwest, for a distance of 340 ft. (103.6 m.). This point must have been practically at the edge of the salt mass, for when it was reached the working face caved and water and sand rushed in, filling the entry and shaft to sea level within a few hours. A second shaft was soon started, but the difficulty of penetrating a considerable thickness of quicksand at 210 ft. (64 m.) led to its abandonment a few feet short of reaching the salt body. An attempt was also made to extract the salt by pumping it out as brine. In this manner several thousand tons of salt were obtained, but, owing to the fact that the salt in this part of the island is much impregnated with oil and gas, it was found difficult to crystallize it, and it was also left with rusty stains. The unconsolidated character of the overburden also permitted the escape of brine to the surface, which, in spots, threatened to cave; hence it soon became evident that this process would result in reducing the whole island to sea level and salt mining operations were therefore abandoned.

Harris<sup>5</sup> says that the Gulf company put down another series of 10 test wells, designated A to J. He gives the logs of several of these wells, one of which (well I) is of considerable geologic interest, but unfortunately neglects to give their exact locations. The locations of wells H, I, and J on the accompanying map (Fig. 1) are therefore only approximate. I quote the log of well I.

*Notes on Well I, Belle Isle (Location No. 12 of Knapp's Notes)*

|                                     | Feet |
|-------------------------------------|------|
| Hard rock, one-third sulphur .....  | 203  |
| Salt .....                          | 375  |
| Salt, limestone, and oil sand ..... | 430  |
| Quicksand .....                     | 450  |

<sup>5</sup> G. D. Harris: *Louisiana Geological Survey, Bulletin No. 7* (1907), 18-26.

*Notes on Well I, Belle Isle (Location No. 12 of Knapp's Notes).—(Continued)*

|   | Feet |
|---|------|
| Sulphur and hard shale.....   | 485  |
| Sulphur rock.....   | 503  |
| Sand rock with showings of gas and oil.....   | 550  |
| Salt.....   | 585  |
| Hard material, no salt.....   | 655  |
| Sand rock, showings of gas and oil.....   | 675  |
| Improving showings of gas and oil.....  | 700  |
| Flint rock of the hardest kind.....   | 735  |
| Salt.....   | 840  |
| Hard material.....  | 856  |
| Material softer, good showing.....  | 1010 |
| Gas pressure; brown sand rock and blue shale containing oil; gas flame 25 ft. high..... | 1165 |
| Loose brown oil sand and slate.....   | 1202 |
| Bottom of 4-inch pipe.....  | 1210 |
| Gas blowout; oil showing improving.....   | 1230 |
| Increase in gas.....  | 1375 |
| Diamond drill installed.....  | 1430 |
| Rock salt.....  | 1455 |
| Iron sand; probable depth from sample left in bottle.....                               | 1550 |
| Gas blew water out of well; oil showing on slush tank.....                              | 1575 |
| Salt.....   | 1625 |
| Strong gas pressure; better oil showing.....  | 1945 |
| Salt, with showing of gas and oil to bottom.....  | 2035 |
| Total depth.....  | 2359 |

*Partial Section of New Orleans Mining and Milling Co.'s Well No. 3  
(Location No. 10 of Knapp's notes)*

|   | Feet |
|---|------|
| First oil showing.....                        | 1396 |
| Oil showing increasing.....                   | 1506 |
| Big gas showing; good oil showing.....        | 1584 |
| Small gas showing.....                        | 1786 |
| Small gas showing.....                        | 1930 |
| Gas and oil in sand.....                      | 2190 |
| Last 80 ft. give pronounced oil showing.....  | 2270 |
| Biggest pyrite showing; good oil showing..... | 2389 |
| Total depth.....                              | 2411 |

No salt reported in this well.

In 1906, the New Orleans Mining & Milling Co., having taken over the property, drilled two wells near the north end of the island, in an effort to find oil. Although the deeper of these wells reached 2411 ft. (734.9 m.) and is less than 1000 ft. (304.8 m.) north of the old salt mine shaft, it found no salt, indicating that the northern face of the dome is very abrupt. This well reported good shows of gas and oil between 1396 and 2389 ft. (425.5 and 728 m.), but apparently no commercial accumulations were encountered.

## Section of Knapp Well No. 1, Belle Isle

|  | Feet           |
|--|----------------|
| Shale of varying hardness; at 75-80 ft. went through old mine timber; at 100 ft. pyrites showed.....   | 0- 114         |
| Whitish limestone.....   | 115- 122       |
| Shale.....   | 122- 128       |
| Gravelly shale.....  | 128- 133       |
| Hard lime.....   | 133- 142       |
| Oil sand; oil at 143 ft.....   | 142- 145       |
| Shale and specks of sulphur.....   | 145- 159       |
| Hard streak of lime; gas from 150 to 160 ft.....   | 159- 160       |
| Shale, showing oil.....  | 160- 186       |
| Blue clay; <i>Rangia cuneata</i> , fish bones, and lignite at 186-189 ft.....  | 186- 187       |
| Rotten limestone.....  | 187- 200       |
| Probably salt.....   | 200- 211       |
| Salt of varied character.....  | 211- 912       |
| Salt with strong oil odor and taste to 700 ft.   |                |
| Salt, impure, with anhydrite, 418-767 ft.  |                |
| Puff of gas, 838 ft.   |                |
| Anhydrite, 844, 864 ft.  |                |
| Salt, varied.....  | 912-1790       |
| Slight bubbling of gas at 1090 ft.   |                |
| Gas enough to give flame 2 or 3 ft. above 10-in. pipe at 1348 ft.; this gas has a strong odor of crude oil; does not react for $H_2S$ with acetate of lead; came from salt dirty with black shale; similar gas at 1423 ft.; both the puffs of gas lasted for a couple of days and then died out. |                |
| Clear, clean salt at 1425-1790 ft.   |                |
| Greenish shale; mixed salt, gypsum incrusting salt, also black sand at 1800 ft.....  | 1790-1800      |
| Impure gypsum; gas held in salt crystals, which give a crackling noise when dissolving.....  | 1800-1806      |
| Gas sand.....  | 1806-1810      |
| Impure gassy salt.....   | 1810-1840      |
| Soft gypsum.....   | 1840-1850      |
| Impure salt showing a little gas and oil.....  | 1850-1904      |
| Probably impure salt; overflow frothing with gas and showing oil.....  | 1906-1965      |
| White salt.....  | 1965-2030      |
| White salt with gas and oil showing; dark amber oil at 2100 ft.....  | 2030-2112      |
| Bits of pyrite and lime.....   | 2112-2114      |
| Opaque salt with puffs of gas; fine amber oil obtained by straining scum at 2190 ft.....   | 2114-2190      |
| Salt, varied, clear and white or opaque and dirty with gas; oil obtained by straining froth at 2500 ft.....  | 2190-2520      |
| Salt, particles of galena and pyrites of iron and copper.....  | 2520-2606      |
| Salt, anhydrite, and sulphur.....  | 2606-2628      |
| Salt, impure and crystal; varying hardness; good oil showing; puff of gas at 2722 ft. threw water up 20 ft.....  | 2628-2740      |
| Black sand—magnetite.....  | 2740-2745      |
| Gas throwing water up 40 ft. in derrick.....   | 2783           |
| Dark-red oil in salt.....  | 2900           |
| Anhydrite.....   | 2940-3005      |
| Bottom of well very hard—anhydrite or gypsum.....  | 3005-3171-3171 |

*Notes on Knapp Well No. 2, Belle Isle*

|   | Feet    |
|---|---------|
| Sand, gravel, clay, streaks of limestone to sulphur rock..... | 0-290   |
| Gypsum and sulphur rock.....                                  | 290-390 |
| Sulphur, gypsum, and anhydrite, very porous.....              | 390-880 |
| Salt.....   | 880-890 |

Remarks: Average sulphur, 7.5 per cent.; best sample, 35 per cent.; slight showing of oil at 390 ft., where porous rock began. Well was 430 ft. southwest of first and about 100 ft. southeast of the little blacksmith shop, in flat back of salt works.

*Log of Knapp Well No. 3, Belle Isle*

|  | Feet    |
|--|---------|
| Stiff clays.....   | 0-127   |
| Streaks of sand, clay shale, gravel, wood, and wood fiber..... | 127-460 |
| Lime rock.....   | 460-475 |
| Gypseous material, clay, and salt.....                         | 475-550 |
| Impure salt.....   | 550-620 |
| Clear salt.....  | 620-800 |

Not a particle of oil or gas showed; nor any sulphur.

In 1907-08, I. N. Knapp drilled three wells on the island, the northernmost of which, No. 1, close to shaft No. 1 entered salt at about 140 ft. (42.6 m.) and continued in it to 3050 ft. (929.6 m.), though in the last 600 ft. (182.9 m.) several occurrences of limestone, sand, anhydrite, sulphur and oil were noted. This well also found a canary yellow oil of 37° Bé gravity at less than 125 ft. (38 m.) It encountered heavy puffs of gas at many horizons in the salt, and throughout the depth of 1500 to 3171 ft. (457 to 966 m.) found dark red oil of 37° Bé in salt and stopped in hard bottom. For the last 300 ft. (91 m.), this well labored very hard on account of heavy gas pressure, and was abandoned at the depth of 3171 ft.

In 1916-17, the writer drilled six wells on the island for the purpose of ascertaining the extent of the sulphur deposits known to overlie the salt in places. This work has demonstrated that the sulphur-bearing rock is very impure and is confined chiefly to a narrow belt across the northern part of the island, though it has also brought to light some very interesting information on the configuration and extent of the salt surface. It became evident, however, that as a potential source of sulphur Belle Isle may be eliminated.

*Topography*

Belle Isle consists of a ridge about  $\frac{3}{4}$  mile long, extending in a north-east-southwest direction, and rising to a height of 80 ft. above the surrounding low marsh (see Fig. 1). The ridge is cut by a pronounced saddle which practically isolates a small hill at the northeastern extremity, known locally as Bald Hill. This depression is of especial interest, for it appears to be a reflection of the contour of the salt mass itself.

### Geology

The surface of the island is covered by unconsolidated Recent sediments, the surficial layer being in general a clay. This is commonly underlain by a thick mass of sand, but the beds are highly lenticular and in some localities the sand is irregularly interbedded with lenses of clay, gravel and shale. Limestone with the pores and cracks filled with sulphur is reported in certain wells; also gypsum and anhydrite. At the northern end of the island the surface deposits are inclined at an angle considerably more than that of the surface of the underlying salt.

The most detailed section extant of the strata above the salt is that measured by Veatch<sup>6</sup> at the old salt mine shaft and reproduced below. The occurrence of clay containing barite and metallic sulphides is of especial interest, and it is also noteworthy that the upper surface of the salt mass is not sharply defined but grades up through impure salt to "clay with large salt crystals."

#### Section of Shaft No. 1, by A. C. Veatch

| No. | Depths     | (Elevation above tide 7 ft. (2.1 m.))  | Feet | Inches |
|-----|------------|--|------|--------|
| 1   | 0 - 4      | Clay.....  | 4    | 0      |
| 2   | 4 - 13     | Hard sand.....   | 9    | 0      |
| 3   | 13 - 30    | Blue clay.....   | 17   | 0      |
| 4   | 30 - 40    | Blue clay and sand.....  | 10   | 0      |
| 5   | 40 - 63    | Hard clay and gravel.....  | 23   | 0      |
| 6   | 63 - 68    | Blue clay with crystalline masses, from the size of marble to a man's head, of barite, galena, sphalerite, pyrite and chalcopryrite..... | 5    | 0      |
| 7   | 68 - 95    | Blue clay and shells.....  | 27   | 0      |
| 8   | 95 - 96½   | Rock. Impure black limestone; and barite.....  | 1    | 6      |
| 9   | 96½ - 103  | Blue clay with masses of barite near the base.....   | 6    | 6      |
| 10  | 103 - 116  | Dark-colored clay with large salt crystals.....  | 14   | 0      |
| 11  | 116 - 117  | Dark-colored clay with oil.....  | 1    | 0      |
| 12  | 117 - 142  | Salt with dark-colored clay.....   | 25   | 0      |
| 13  | 142 - 162  | Discolored salt.....   | 35   | 0      |
| 14  | 162 - 163  | White limestone.....   | 0    | 8      |
| 15  | 163 - 175* | Dirty salt becoming white.   |      |        |

\* This was the depth of the shaft at the time of Mr. Veatch's departure, May 19, 1899.

Further data on the character of the material above the salt are given in the following logs of the six wells drilled by the writer, and of the Knapp No. 2 well.<sup>7</sup>

<sup>6</sup> C. A. Veatch: *Louisiana Geological Survey Report* (1899), 221-229.

<sup>7</sup> *Op. cit.*

*Logs of Syndicate Wells Drilled at Belle Isle, Louisiana, December, 1916, to March, 1917*

*Well No. 1*

| From | To<br>(In Feet) | Thickness | Strata                                   |
|------|-----------------|-----------|--|
| 0    | 13              | 13        | Surface muck and clay                    |
| 13   | 102             | 89        | Clays and shales with a little sand      |
| 102  | 153             | 51        | Sanded red and blue clays and shales     |
| 153  | 224             | 71        | Gray sand                                |
| 224  | 254             | 30        | Hard gray sand                           |
| 254  | 260             | 6         | Very hard gray sand (probably sandstone) |
| 260  | 289             | 29        | Soft gray sand                           |
| 289  | 291             | 2         | Hard shale                               |
| 291  | 292             | 1         | Gravel                                   |
| 292  | 330             | 38        | Fine gray sand                           |
| 330  | 355             | 25        | Gravel                                   |
| 355  | 375             | 20        | Coarse sand                              |
| 375  | 389             | 14        | Coarse sand (very hard).                 |
| 389  | 419             | 30        | Hard blue rock                           |
| 419  | 438             | 19        | Sand                                     |
| 438  | 441             | 3         | Rock                                     |
| 441  | 457             | 16        | Sand                                     |
| 457  | 465             | 8         | Shale                                    |
| 465  | 481             | 16        | Sandstone at first turning to limestone  |

*Well No. 2*

| From | To<br>(In Feet) | Thickness | Strata   |
|------|-----------------|-----------|--|
| 0    | 20              | 20        | Surface muck and clay                                  |
| 20   | 38              | 18        | Fine gray sand   |
| 38   | 56              | 18        | Banded red and blue clays                              |
| 56   | 80              | 24        | Fine gray sand   |
| 80   | 127             | 47        | Clay and shale   |
| 127  | 135             | 8         | Sandstone  |
| 135  | 210             | 75        | Sand   |
| 210  | 225             | 15        | Hard sand  |
| 225  | 246             | 21        | Clay and shale   |
| 246  | 281             | 35        | Gray sand with a little oil                            |
| 281  | 286             | 5         | Limestone with some gypsum                             |
| 286  | 341             | 55        | Sulphur rock with pores and cracks filled with sulphur |

*Logs of Syndicate Wells Drilled at Belle Isle, Louisiana, December, 1916,  
to March, 1917.—(Continued)**Well No. 3*

| From | To<br>(In Feet) | Thickness | Strata                                   |
|------|-----------------|-----------|--|
| 0    | 80.00           | 80.00     | Surface muck and clay with a little sand |
| 80   | 163.00          | 83.00     | Blue and red banded clays and shales     |
| 163  | 173.00          | 10.00     | Soft sand                                |
| 173  | 224.00          | 51.00     | Hard blue clay (gumbo)                   |
| 224  | 280.00          | 56.00     | Sand                                     |
| 280  | 295.00          | 15.00     | Rock                                     |
| 295  | 310.00          | 15.00     | Sand                                     |
| 310  | 375.00          | 65.00     | Gravel                                   |
| 375  | 380.00          | 5.00      | Hard clay                                |
| 380  | 386.00          | 6.00      | Gravel                                   |
| 386  | 391.00          | 5.00      | Rock                                     |
| 391  | 396.00          | 5.00      | Sulphur rock                             |
| 396  | 399.00          | 3.00      | Clay                                     |
| 399  | 404.00          | 5.00      | Gravel                                   |
| 404  | 445.00          | 41.00     | Sand                                     |
| 445  | 566.25          | 121.25    | Gravel and rock salt                     |

*Well No. 4*

| From | To<br>(In Feet) | Thickness | Strata   |
|------|-----------------|-----------|--|
| 0    | 10              | 10        | Surface muck and clay                                |
| 10   | 85              | 75        | Clay   |
| 85   | 198             | 113       | Banded blue and red clay and shale                   |
| 118  | 225             | 27        | Hard blue clay                                       |
| 225  | 405             | 180       | Blue sand  |
| 405  | 415             | 10        | Sandstone (?)  |
| 415  | 465             | 50        | Gray sand with some gravel                           |
| 465  | 489             | 24        | Gravel and sand (chert pebbles)                      |
| 489  | 494             | 5         | Rock   |
| 494  | 499             | 5         | Sulphur rock   |
| 499  | 513             | 14        | Gravel mostly chert                                  |
| 513  | 525             | 12        | Rock   |
| 525  | 565             | 40        | Sand, shale and gravel                               |
| 565  | 715             | 150       | Gypsum with sulphur (last 35 ft. sulphur sparingly). |



*Logs of Syndicate Wells Drilled at Belle Isle, Louisiana, December, 1916,  
to March, 1917.—(Continued)*

*Well No. 5*

| From | To<br>(In Feet) | Thickness | Strata                                     |
|------|-----------------|-----------|--|
| 0    | 6               | 6         | Surface muck and clay                      |
| 6    | 179             | 173       | Fine sand                                  |
| 179  | 315             | 136       | Sand, gravel and shale (some gas in shale) |
| 315  | 319             | 4         | Shale                                      |
| 319  | 329             | 10        | Hard clay                                  |
| 329  | 355             | 26        | Shale (some gas)                           |
| 355  | 393             | 38        | Gravel and hard clay                       |
| 395  | 430             | 37        | Gravel                                     |
| 430  | 446             | 16        | Shale                                      |
| 446  | 490             | 44        | Lime                                       |
| 490  | 534             | 44        | Salt                                       |

*Well No. 6*

| From | To<br>(In Feet) | Thickness | Strata                                  |
|------|-----------------|-----------|---|
| 0    | 4               | 4         | Surface muck and clay                   |
| 4    | 415             | 410       | Gray sand (medium fineness)             |
| 415  | 460             | 55        | Lime and gypsum, with some sulphur      |
| 460  | 505             | 45        | Lime, harder and showing little sulphur |
| 505  | 545             | 40        | Salt                                    |

Syndicate well No. 2 was lost at a depth of 341 ft. (103.9 m.), but as it was drilled close to Knapp No. 2, which penetrated to 890 ft. (271.3 m.), it may be disregarded, and Knapp No. 2 used as a criterion. Knapp No. 2 reports 590 ft. (179.8 m.) of gypsum, anhydrite and sulphur immediately above the salt, and Lucas No. 2 passed through 60 ft. (18.3 m.) of sulphur rock at the same horizon. These two wells must have encountered the sulphur deposit at its maximum thickness, for Syndicate wells 3 and 4 found only 5 ft. (1.5 m.) of sulphur rock and Nos. 1, 5 and 6 found no commercial sulphur rock at all. Knapp well No. 1 found a little sulphur at two horizons and the Gulf well I reports a number of sulphur-bearing strata. As shown by the later drilling, however, the sulphur encountered in these wells is either in very thin layers or in local lenses of no commercial importance. The thick deposit in Knapp well No. 2, which Mr. Knapp says averages 7.5 per cent. sulphur, is evidently of very limited extent.

The relation of the sulphur to the salt, and the available information as to the contour of the salt surface itself, are shown by the tabulated well data in the accompanying summary of logs.

*Summary of Logs of Wells Drilled at Belle Isle, 1896-1917*

| Well                      |     | Elevation + | Oil Showings At  | Sulphur Rock At              | Top of Salt | Total Depth | Reference |
|---------------------------|-----|-------------|--|------------------------------|-------------|-------------|-----------|
| Name                      | No. |             |  |                              |             |             |           |
| Lucas.....                | 1   | 5           | .....  | .....                        | .....       | 590         | a         |
| Lucas.....                | 2   | 13          | Traces   | 276                          | 335         | 410         | a         |
| Lucas.....                | 3   | 7           | 110-125  | d110-125                     | 135         | 170         | a         |
| Lucas.....                | 4   | 5           | 135-275  | d135-275                     | 325         | .....       | a         |
| Gulf shaft....            | 1   | 7           | 116-117  | None                         | 117         | 326         | b         |
| Gulf shaft....            | 2   | 5           | None?  | 16.0-190                     | 210         | 196         | b         |
| Gulf shaft....            | H   | 5           | None   | 340-391                      | .....       | 420         | b         |
| Gulf shaft....            | I   | 20          | 303, 940, 985,<br>1086, 1105, 1212,<br>1365, 1370          | 256 to 285, 870,<br>1206     | 1,545       | 2,450       | b         |
| Gulf shaft....            | J   | 3           | None   | None                         | .....       | 600         | b         |
| New Orleans<br>Mining Co. | 2   | ..          | .....  | .....                        | .....       | 1,740       | c         |
| New Orleans<br>Mining Co. | 3   | ..          | 1396, 1506, 1584,<br>2190, 2270, 2389                      | None?                        | .....       | 2,411       | c         |
| Knapp.....                | 1   | ..          | 143, 160, 1850 to<br>1965, 2100, 2190,<br>2500, 2722, 2900 | d145 to 159, 2606<br>to 2628 | 150         | 3,171       | c         |
| Knapp.....                | 2   | ..          | 390  | 290 to 880                   | 880         | 890         | c         |
| Knapp.....                | 3   | ..          | None   | None                         | 550         | 800         | c         |
| Syndicate....             | 1   | ..          | None   | None                         | .....       | 481         |           |
| Syndicate....             | 2   | ..          | 246-281  | 286-341                      | .....       | 341         |           |
| Syndicate....             | 3   | ..          | None   | 391-396                      | 445         | 566         |           |
| Syndicate....             | 4   | ..          | 320-499  | 494-499, d565-715            | .....       | 715         |           |
| Syndicate....             | 5   | ..          | None   | None                         | 490         | 534         |           |
| Syndicate....             | 6   | ..          | None   | d415-505                     | 505         | 545         |           |

a A. C. Veatch: *Louisiana Geological Survey, Report* (1899), 221-229.

b G. D. Harris: *Louisiana Geological Survey, Bulletin* No. 7 (1907), 18-26.

c G. D. Harris, *U. S. Geological Survey, Bulletin* No. 429 (1910), 43-48.

d Sulphur present in traces only.

These data are shown graphically in the accompanying cross-sections (Fig. 2). It is evident, in the first place, that the salt dome extends much farther to the southeast than has hitherto been supposed, and that it probably underlies all of Belle Isle Lake at a depth of less than 800 ft. (243 m.). This eastern and southeastern portion of the salt body forms, with respect to the highest part of the latter, a kind of terrace or shoulder, dipping only from 490 to 505 ft. (149.4 to 153.9 m.) in a distance of nearly 2000 ft. (609 m.) (between Syndicate wells 5 and 6). Beneath the ridge and hills of Belle Isle the salt is not nearer the surface; on the contrary, the nearest salt was found in the northern part of the island in the flat area where the first shaft was dug, where salt was found at 113 ft. (34.5

m.). To the north of this shaft, or at the base of the outlying hill forming the northern extremity of the island, the salt pitches off with extreme abruptness, from 113 ft. at the shaft (No. 1) to 1545 ft. (471 m.) in well I.

The most interesting feature, however, is the deep hollow, or saddle, which crosses the island transversely near its northern extremity, and which is apparently reflected in the topographic depression practically isolating the hill near which the old shaft was sunk. As shown by the

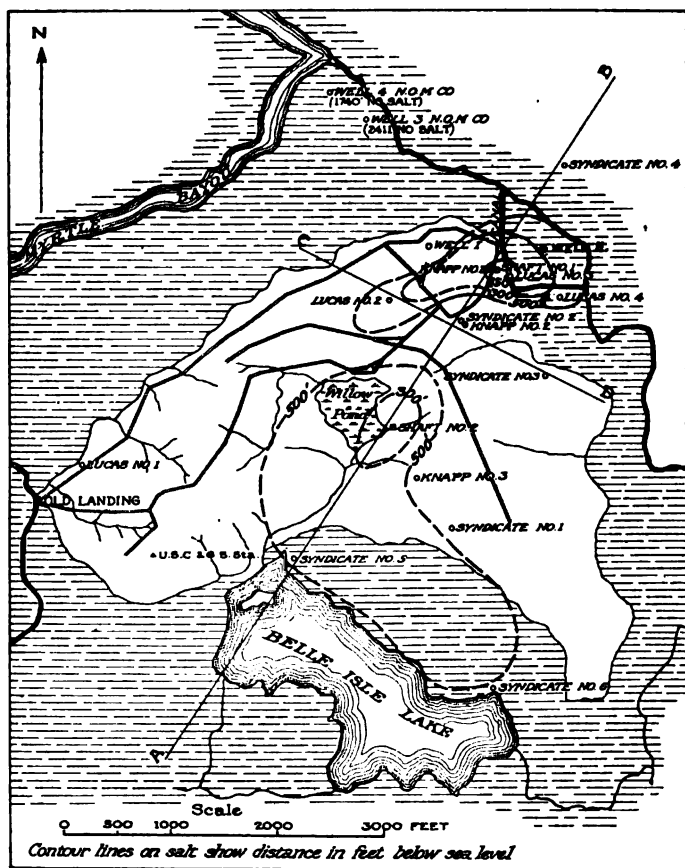


FIG. 1.—MAP SHOWING LOCATIONS OF RECENT WELLS AT BELLE ISLE, LA.

record of the Knapp No. 2 well, this hollow is at least 880 ft. (268.3 m.) deep. On either side of it, as shown by the two shaft sections in Fig. 1, the salt rises nearly to the surface, being found at a depth of only 113 ft. in shaft No. 1, and 210 ft. (64 m.) in shaft No. 2. It is only in this saddle, or notch in the salt surface, that a considerable thickness of the sulphur rock was found. The Knapp No. 2 well found 590 ft. (179.8 m.) of sulphur rock with the salt at 880 ft., and the Lucas No. 1, which is

somewhat higher up on the northwest slope of the saddle, found 60 ft. (18.3 m.) of sulphur rock with salt at 335 ft. (102 m.).

Although slight irregularities in the salt surface have been observed at other salt domes along the Gulf Coast, the writer knows of no case in which the irregularity is nearly as pronounced as that just described. Minor depressions in the salt have been plausibly ascribed to solution by the fresh water that doubtless percolates down to the salt in places, but the saddle or hollow at Belle Isle appears to be of primary rather than of secondary origin. This is suggested by the fact that it is now filled with nearly 600 ft. (182 m.) of hard sulphur rock which was undoubtedly deposited by the escape of sulphur and sulphurated hydrogen of an incipient

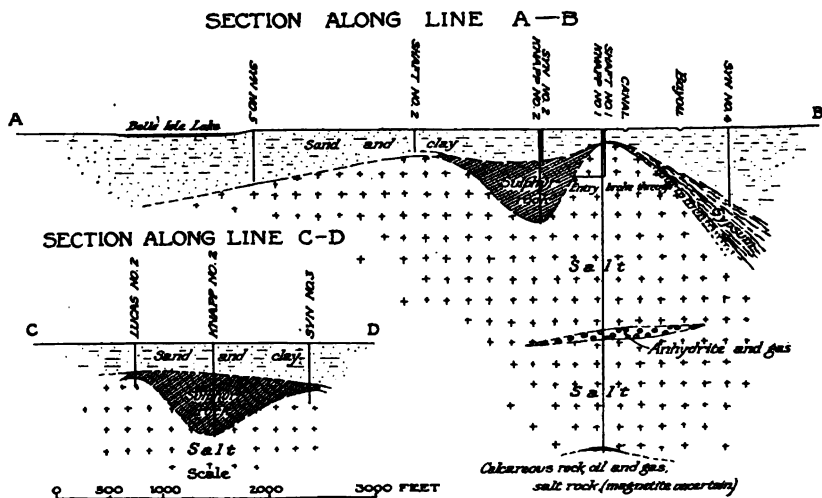


FIG. 2.—CROSS-SECTIONS OF RECENT WELLS SHOWING SULPHUR POCKET AT BELLE ISLE, LA.

volcanic origin, analogous to the fumaroles on Spindle Top, Texas, as pointed out by the writer to Prof. R. T. Hill<sup>8</sup> in his description of the Beaumont oil field, and also as pointed out by Eugene Coste.<sup>9</sup> The same agencies undoubtedly give rise to the salt dome itself, and perhaps also to the marked construction and depression in the topographic ridge above. It seems reasonable to suppose that the Belle Isle dome is of compound origin, and that the salt ascended chiefly along two lines of weakness, the main one located in the southern part of the island and the smaller one beneath the hill at the northern end. It is probable that these masses or lines of weakness unite at no great depth, or that they

<sup>8</sup> R. T. Hill: The Beaumont Oil Field, with Notes on Other Oil Fields of the Texas Region. *Journal of the Franklin Institute* (August–October, 1902), 154, 143, 225, 263.

<sup>9</sup> Eugene Coste: The Volcanic Origin of Natural Gas and Petroleum. *Journal of the Canadian Mining Institute* (1903), 6, 74–128.

represent simply one plane of weakness, along which the escape of the gas was more effectively obstructed beneath the saddle than beneath the two salt knobs. As to the manner in which the salt, with its gas and oil inclusions, attained its present position, at a depth of 1500 to 3171 ft.—whether in solution or as semi-plastic mass—has never been satisfactorily explained. The solution of the problem must await a further understanding of the genetic relations of the oil, gas, sulphur and salt, and such an understanding should result from thorough analytical and experimental observations or from practical demonstration test borings, or, more likely, from a combination of both. The writer urgently suggests the drilling of a deep well, of say 5000 to 7000 ft. (1524 to 2133 m.) or more at a point carefully selected above the fissure in order to arrive at a conclusion more practical than the innumerable hypothetical suggestions now current.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## The Sulphur Deposits in Culberson County, Texas

WILLIAM B. PHILLIPS,\* HOUSTON, TEXAS

(St. Louis Meeting, October, 1917)

THE earliest mention of the sulphur deposits in what is now Culberson County, Texas, seems to be contained in "Explorations and Surveys for a Railroad Route from the Mississippi River to the Pacific Ocean, United States War Department, 1853-54." Captain John Pope (he was not commissioned Captain until 1856) had charge of one of the divisions of the Survey. In his report on the geology of the route, William P. Blake noted the analysis of a greenish earthy powder from the bed of Delaware Creek, at the springs, which contained 18.28 per cent. of free sulphur. The report states:

"From the chemical examination of the waters of the spring, by Dr. Booth, it appears that chloride of sodium is the principal solid ingredient, but considerable quantities of sulphide of sodium are present. \* \* \* \* \* The waters from the springs examined by Dr. Booth were alkaline, but it is possible that some of the springs are acid; and in this case the precipitation of sulphur would ensue on the mixture of the acid waters with those charged with alkaline sulphurets and the origin of the deposits of sulphur would be thus explained."

This is the first reference to the existence of sulphur in that region, the first mention of the character of the water found there and the first suggestion as to the origin of the deposits. The possibility of the existence of acid waters was confirmed by E. M. Skeats, nearly 50 years later.<sup>1</sup> In this publication, Skeats says that the water from the sulphur deposits in Section 11, Block 114, Township 5, contains 79.08 grains of free sulphuric acid per gallon. This locality is about 13 miles (21 km.) south of Delaware Creek and about 35 miles (56 km.) east of the Delaware Springs.

The next published reference to the existence of sulphur in Trans-Pecos Texas is in the First Annual Report of the Geological Survey of Texas, 1889, page 226, where W. H. von Streeruwitz mentions that sulphur is found in that region, but gives no further information. In the

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\* Consulting Engineer.

<sup>1</sup> Sulphur, Oil and Quicksilver in Trans-Pecos Texas. University of Texas Mineral Survey, *Bulletin* No. 2 (1902).

Third Annual Report of that Survey, 1891, in his report on the Geography, Topography and Geology of the Llano Estacado, or Staked Plain, page 165, W. F. Cummins says:

"We had been informed that there was a bed of native sulphur in this vicinity (Castile Springs), but after a diligent search for it for two days were unable to find it."

The Castile Spring to which Cummins referred is about 16 miles northwest of Maverick Springs. There are no known deposits of sulphur in this immediate vicinity, but in Sections 8, 16, 17 and 18, Block 61, a few miles west of Castile Spring, there are some very interesting deposits at and near the Grant mines. These have been prospected to some extent but have not been developed.

The next published account of the sulphur deposits in Culberson County is contained in an article by Dr. Eugene A. Smith, State Geologist of Alabama (*Science*, May 1, 1896). This was republished in full in the bulletin of the University of Texas Mineral Survey above referred to. It was condensed from a private report made by Dr. Smith. The appearance of this bulletin, in 1902, marks the first attempt to describe these deposits in some detail.

It contained the article by Dr. Smith, just alluded to, the full report of E. M. Skeats, who had been engaged by the University Mineral Survey to study the deposits and describe them in as much detail as was then possible, and the results of personal observations by the writer, who was at that time Director of the Survey, and whose acquaintance with these deposits has been continued since 1901.

In 1904, by coöperation between the U. S. Geological Survey and the Mineral Survey, George B. Richardson published the results of his observations in that region. His report,<sup>1</sup> which contained a geological map and was a valuable contribution to the literature concerning the district, has been out of print for some time. There are still a few copies of the map, which may be obtained from the Bureau of Economic Geology, University of Texas.

In 1912,<sup>2</sup> 1916,<sup>3</sup> and 1917<sup>4</sup> the writer published articles on the subject of sulphur in Culberson County.

These furnish the principal published data concerning the district, but we understand that a bulletin on this subject, by E. L. Porch, Jr., Bureau of Economic Geology, University of Texas, will soon be available.

<sup>1</sup> Reconnaissance in Trans-Pecos Texas North of the Texas and Pacific Railway. *Bulletin No. 9, University of Texas Mineral Survey* (1904).

<sup>2</sup> Sulphur Deposits in Culberson County. *American Fertilizer* (1912), 36, No. 12, 449 (1912).

<sup>3</sup> The Sulphur Situation in Culberson County, Texas. *Manufacturers' Record*, Baltimore (Dec. 7, 1916).

<sup>4</sup> Development of Sulphur Deposits in Culberson County, Texas. *Manufacturers' Record*, Baltimore (July 19, 1917)..



Many private reports have been consulted through the kindness of the persons for whom they were made, and the writer has availed himself of his own copious notes made during a period of 16 years.

### *Location of the Sulphur Districts*

A few years ago the Legislature of Texas divided El Paso County, forming Culberson County from the eastern portion. This accounts for the early references to sulphur in El Paso County. The principal sulphur districts are from 35 to 40 miles (56 to 64 km.) north of the town of Toyah, Reeves County, and a little further from Pecos, the county seat. These towns are on the Texas & Pacific Railroad west of the Pecos River. The Santa Fe lines also run into Pecos from the north, through Carlsbad and Roswell to Clovis, where connection is made with the main lines of this system.

There are many outlying and separate localities where more or less prospecting has been done, but the principal development has been in the Maverick Springs district, from 15 to 20 miles west of the Santa Fe lines at Orla, Reeves County, 40 miles north of Pecos. The first attempts at mining and extracting the sulphur from any of these deposits were made about 20 years ago by a Mr. Choteau, St. Louis, on what is known as the Cooksey property, in Sections 10 and 15, Block 114. A vertical extractor, for use of steam, was in use there for some time and it is reported that two car loads of sulphur were shipped to St. Louis. Of recent times, however, most of the prospecting and development work has been carried on in the Maverick Springs district, about 15 miles southeast of the Cooksey property. The deposits at and near Maverick Springs have been known for more than 20 years. The old Kyle property (West Texas Sulphur Co.), on which there was a sort of extractor and an attempt to build a calcarone, is within a mile of the Springs, the Dot property is 3 miles to the southwest (the Toyah Valley Sulphur Co. owns a good deal of land near the Dot and also near the old Kyle), while the holdings of the United States Sulphur Co. and the Michigan Sulphur & Oil Co. are toward the head of Virginia Draw, about 5 miles southwest of Maverick Springs.

The properties that up to this time have undergone some prospecting and development in the Maverick Springs district are: The West Texas Sulphur Co., the Toyah Valley Sulphur Co., the Sun Sulphur Co., the United States Sulphur Co. and the Michigan Sulphur & Oil Co. All of these are comprised within an area 5 miles southwest of Maverick Springs.

The American Sulphur Co. is opening a property near Rustler Springs and has produced a small amount of sulphur.

There are other localities in the county where discoveries have been made and where more or less prospecting has been done. The principal ones are: Block 42, Sections 7, 8, and 9, Pat, Lone Wolf and Golson

claims; Block 60, Sections 15 and 16; Block 61, Sections 8, 16, 17 and 18, Grant mines, etc.; Block 109, Sections 4, 5, 9, 10 and 24, Nelson claims, etc.; Block 113, Section 14, the Hicks claims; Block 114, Sections 10 and 15, the Cooksey property; Block 115, Sections 10, 11, 18, 19, 22 and 23. In Reeves County, Block 59, Section 17 or 9, near the old Huling-Ross oil wells and within a short distance of the oil well bored to a depth of 4100 ft. (1249 m.) by The Texas Co.

With the exception of the deposits in the Maverick Springs district and the one last mentioned, none of the discoveries is within 25 miles of a railroad.

### *Geology*

The Guadalupe Mountains, 45 miles (72 km.) northwest of Maverick Springs, form the chief topographic feature of this entire region. They attain, in El Capitan, an elevation of 8690 ft. (2649 m.), the highest point within the State. Thirty miles southeast of these mountains and separated from them by a rolling plain are the Rustler Hills, which are much lower. From the Rustler Hills east to the Pecos River, the slope is gradual but constant, the elevation at Orla, on the Santa Fe lines, being 2853 ft. (869 m.).

George G. Shumard<sup>4</sup> considered the Guadalupe Mountains as Permian. Jules Marcou<sup>5</sup> and Robert T. Hill<sup>6</sup> classed them as Carboniferous. R. S. Tarr<sup>7</sup> says:

"No Permian beds appear between the Guadalupe Mountains and the Pecos River in the section studied, but wherever the Carboniferous is covered by later formations these deposits are either Quaternary or Cretaceous."

Walter P. Jenney<sup>8</sup> speaks of the Guadalupe Mountains as Carboniferous. George B. Richardson<sup>9</sup> considers the Delaware Mountains (the extension of the Guadalupe Mountains to the south) as Permian and is inclined to place the sandstones, limestones and black limestones in the Guadalupe as also in the Permian. In this view he is supported by G. H. Girty, whom he quotes.

On the geological map accompanying his report, Richardson marks an area of some 16 miles (25.7 km.) east of the Delaware Mountains as belonging to the Capitan limestone division of the Permian series of the Carboniferous. East of this area again and for some 16 miles he marks

<sup>4</sup> *Transactions, St. Louis Academy of Science* (1858), 273.

<sup>5</sup> *Geology of North America*.

<sup>6</sup> *Physical Geography of the Texas Region* (1900).

<sup>7</sup> Reconnaissance of the Guadalupe Mountains. *Bulletin No. 3, Geological Survey of Texas* (1892), 21.

<sup>8</sup> *American Journal of Science*, 3d Ser. (January, 1874), 7, 25-28.

<sup>9</sup> Reconnaissance in Trans-Pecos Texas. *Bulletin No. 9, University of Texas Mineral Survey* (1904).

the formation as Castile gypsum, questionably belonging to the Permian series of the Carboniferous. This brings one to the Rustler Hills. These he classes with the Castile gypsum, questionably Permian series of the Carboniferous. But Tarr says that no Permian beds occur between the Guadalupe Mountains and the Pecos River, in the section studied, and it would have been difficult for him to have reached the Pecos from the Guadalupe Mountains without crossing the Rustler Hills, which appear to belong among the transition beds between the true Carboniferous and the true Permian, but it is not yet known to which of these formations they are closest kin.

East of the Rustler Hills and coming into and across the sulphur deposits in the Maverick Springs district are the deposits that Richardson

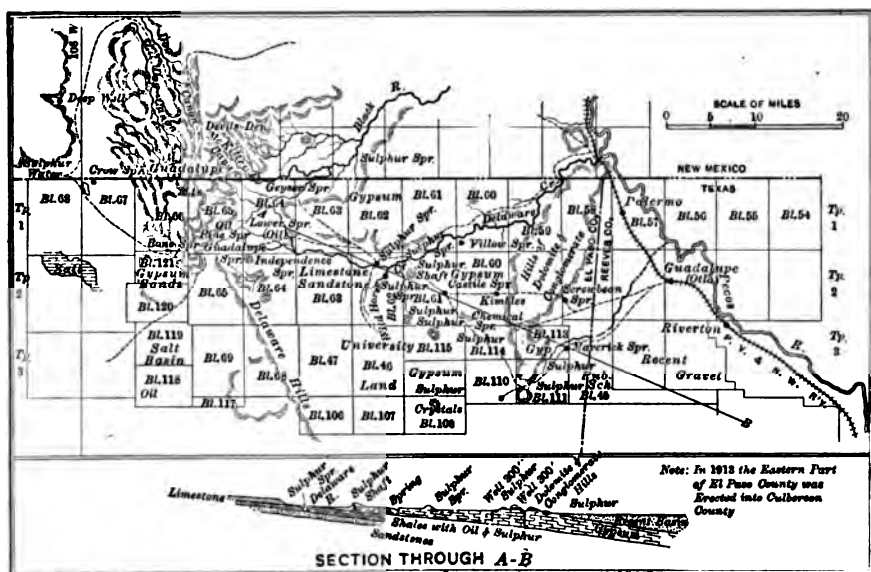


FIG. 1.

classes as Quaternary, Recent and Pleistocene. They are characterized by heavy deposits of gypsum, a few outcrops of "Red Beds" and the remains of dolomite hills. These "Red Beds" are more extensive than Richardson indicates. Instead of occurring over a few acres they are to be observed, at intervals, over a considerable area, especially in Sections 13, 14 and 24, Block 113, and in Section 1, Block 111.

An interesting occurrence in connection with these "Red Beds" was revealed by core drilling a little northeast of the center of Section 1, Block 111, conducted by the writer in November, 1916. This drilling was started in an outcrop of the "Red Beds" which could be traced from

the Virginia Draw eastward to the top of the divide between this Draw and the Maverick Draw. To a depth of 47.5 ft. (14.48 m.) the boring was in red sandstone, which was characterized by many whitish-gray splotches. At 47.5 ft., a bed of massive white gypsum was encountered and in this gypsum were scattered crystals of sulphur. This gypsum was 2.5 ft. (0.76 m.) thick, with red sandstone underneath, to a depth of 74 ft. (22.5 m.) when the boring was discontinued. There was no water in this hole.

On both sides of Virginia Draw and even in the Draw itself are outcrops of a dolomite closely resembling the dolomite that Richardson mentions as occurring in Horseshoe Draw, 2 miles to the northwest. Some of the borings in the upper part of Virginia Draw stopped in dolomite after passing through notable beds of sulphur, and sulphur has been found in the dolomite itself and underneath it. Considering the occurrence of this dolomite in immediate association with the sulphur deposits, I am disposed to regard them, in the Maverick Springs district, at least, as of the same age as the Rustler Hills, *i.e.*, Permo-Carboniferous. The sulphur does occur with rounded gravel, etc., of the Quaternary, but this gravel is evidently not in place, in many localities, and the sulphur associated with it may be due to deposition from gases and alkaline sulphides.

As a further contribution to our knowledge of the geology of the immediate region there is given here the log of a well bored to the depth of 2265 ft. (690 m.), near the south line of Survey 24, Block 110. Two wells were bored here by The Texas Petroleum Co., in 1914 and 1915, but one of them was carried down only to a depth of 541 ft. (164.9 m.). These are known as the Rustler Springs wells and I am indebted to Mr. C. R. Troxel, Toyah, for permission to use the log here given. The locality is about 4 miles (6 km.) southwest of the sulphur property of the Michigan Sulphur & Oil Co. and the holdings of the United States Sulphur Co., toward the head of Virginia Draw. It is between the sulphur deposits that are south and west of Rustler Springs and those that are northeast of these Springs, being about equidistant from the two, *viz.*, about 4 miles. The wells are in the Rustler Hills at an elevation of 250 to 300 ft. above the deposits in Virginia Draw. They go far below the sulphur horizon, as now known, the bottom of the deeper well being about 2000 ft. (609 m.) below the level of Virginia Draw. No deposits of sulphur were found in either well, but several barrels of oil were reported from a depth of 2018 to 2020 ft.

Description of samples from Troxel No. 2 well, Rustler Springs, Culberson County, Texas. Located near the south line of Survey 24, Block 110, Public School Lands, close to Troxel No. 1. Drilled in 1915. Samples examined by Dr. J. A. Udden, Bureau of Economic Geology, University of Texas.

Red Sandy clay 85 ft.

| From | To  | Feet | Formation   |
|------|-----|------|---|
| 85   | 150 | 65   | Greenish-gray and some red clay. Some fragments of gray, compact dolomite.  |
| 150  | 235 | 185  | Dark gray and brownish-gray limestone of very compact texture. Gypsum, quartz grains, fragments of chert and other quartz and pyrite noted. Little gas.   |
| 235  | 240 | 5    | Gray dolomite of very fine texture. Pyrite, gypsum, rounded clear quartz grains and quartz crystals noted.  |
| 240  | 250 | 10   | Dark gray dolomite of fine texture. Effervescence in acid is extremely slow. With this is some marl. Fragments of gypsum, many. There are some rounded grains of quartz and some quartz crystals. Much pyrite. No bituminous fumes noted.   |
| 250  | 255 | 5    | White gypsum of moderately coarse crystalline texture. The sample contains some double pyramidal crystals of quartz, the largest 0.75 mm. in length, with dark transparency.  |
| 255  | 350 | 95   | Mostly gypsum, with some anhydrite, white and gray, mostly moderately coarsely crystalline.   |
| 350  | 405 | 55   | Mostly gray and brownish dark gray dolomite of fine compact texture. In thin section it is seen to contain some anhydrite and is impregnated with pyrite, which occurs in exceedingly fine particles. The sample contains coarse rounded quartz sand and fragments of quartz pebbles. Much anhydrite present. |
| 405  | 450 | 45   | Compact gray dolomite and white anhydrite. Bright yellow pyrite noted.  |
| 450  | 500 | 50   | Anhydrite and gray compact dolomite. Bright yellow pyrite and fragments of quartz noted.  |
| 500  | 515 | 15   | White and gray anhydrite.   |
| 515  | 540 | 25   | White anhydrite with some rounded quartz sand grains, pyrite noted.   |
| 540  | 545 | 5    | White anhydrite.  |
| 550  | 555 | 5    | White and gray anhydrite, some quartz sand, fragments of gray flint, crystals of gypsum, and pyrite.  |
| 555  | 560 | 5    | Anhydrite with some sand, some pyrite and considerable chert which seems to have been in the form of pebbles. Most of the chert is dark. Oil colors on slush.   |
| 560  | 600 | 40   | White anhydrite, some flint present.  |
| 600  | 625 | 25   | White and gray anhydrite. Some quartz, some pyrite and some flint present.  |
| 625  | 630 | 5    | White anhydrite, with a few particles of calcareous material (description by E. L. Porch, Jr.). Gray dolomitic limestone containing pyrite, anhydrite and sand.   |
| 630  | 650 | 20   | Light gray dolomitic limestone containing anhydrite, pyrite and sand. Other sample consists of anhydrite.   |
| 650  | 680 | 30   | Light gray anhydrite containing sand, pyrite and black minerals. Gave off a faint odor of bitumen when heated. An endothyra was noted. Oil colors on slush.   |

| From            | To              | Feet | Formation  |
|-----------------|-----------------|------|--|
| 680             | 715             | 35   | Anhydrite containing a little sand.  |
| 715             | 735             | 20   | Anhydrite containing considerable calcareous material and some sand. On heating, a faint odor of bitumen was given off and a little sulphur deposited in the tube. Label says "gas."                                       |
| 735             | 740             | 5    | Gray sandy limestone containing pyrite, and fragments of pebbles of gray and dark chert, and gypsum. Endothyra noted. Gave off odor of bitumen on heating.   |
| 745             | 757             | 12   | Gray limestone containing sand most of which is below $\frac{1}{8}$ mm. Chert, anhydrite, and pyrite. On heating a little oil distilled off.   |
| 757             | 795             | 38   | Light gray dolomitic limestone containing sand, fragments of chert pebbles, pyrite and anhydrite. Gave off odor of bitumen on being heated. Little gas at 786. At 791 a little water, strong with sulphur.                 |
| 795             | 800             | 5    | Gray anhydrite, and quartz sand, some calcareous material and some dark chert grains.  |
| 800             | 810             | 10   | Anhydrite, white, gray and yellow.   |
| 810             | 822             | 12   | Anhydrite and very dark limestone.   |
| 822             | 828             | 6    | Mostly white and gray anhydrite. Some very dark limestone, some sand and some fragments of greenish shaly rock.  |
| 828             | 837             | 9    | Yellowish and dark limestone with sand and very fine gravel. Pyrite and anhydrite noted. Some foraminifera like a Textularia with perforate walls was observed, and an Anomalina, well preserved.                          |
| 838             | 843             | 5    | Limestone and anhydrite, both in very dark and in very light gray shades. Some sand present. Foraminifera of the types of Globigerina and Textularia quite common.   |
| 843             | 857             | 14   | Anhydrite, mostly white, and some dolomite. Globigerina, Textularia and a foraminifera like Anomalina noted. Pyrite noted.   |
| 857             | 900-<br>943     | 83   | Light gray limestone containing chert, sand pyrite and considerable anhydrite. Gave off the odor of bitumen on being heated. Endothyra noted. Label said "little gas and oil."   |
| 940-<br>943     | 1,030           | 90   | Anhydrite containing calcareous material and a little sand. Some sulphur was deposited in the tube on heating.   |
| 1,030           | 1,152-<br>1,154 | 90   | Light gray limestone containing anhydrite and pyrite. Upon being heated, a little sulphur was deposited in the tube, and a faint odor of bitumen was given off. An Endothyra was noted. Label said "gas and a little oil." |
| 1,152-<br>1,154 | 1,942           | 790  | Gray and dolomitic limestone containing anhydrite, pyrite, sand and a black mineral, probably chert, considerable oil was distilled off in heating the material.   |
| 1,942           | 1,956           | 14   | Gray cherty limestone containing sand and anhydrite. Some oil was distilled off on heating the material.   |

| From  | To    | Feet | Formation  |
|-------|-------|------|--|
| 1,956 | 1,962 | 6    | Gray argillaceous dolomitic sandstone containing a considerable quantity of black mineral, probably chert. Most of the sand is below $\frac{1}{8}$ mm. in size. Considerable oil was distilled off on heating the material.          |
| 1,962 | 1,982 | 20   | Gray dolomitic limestone containing black fragments (probably carbonaceous material), pyrite, sand, chert and anhydrite. A little oil distilled off on heating.  |
| 1,982 | 1,988 | 6    | Gray sandy marl, containing pyrite and black mineral. Considerable oil was distilled off on heating.   |
| 1,988 | 1,994 | 6    | Gray limestone containing considerable chert and sand. Some oil distilled off on heating.  |
| 1,994 | 2,006 | 12   | Dolomitic sandstone containing pyrite and a black mineral. Most of the sand is below $\frac{1}{8}$ mm. Gave off a little oil on heating.   |
| 2,006 | 2,010 | 4    | Sandy dolomite containing pyrite. Considerable oil distilled off on heating.   |
| 2,010 | 2,018 | 8    | Sand and clay (oil sand) containing a black mineral (probably chert). A little pyrite, a very little calcareous material. Gave off considerable oil on heating. Most of the sand is below $\frac{1}{8}$ mm. Endothyra was noted (?). |
| 2,018 | 2,020 | 2    | Several barrels of oil were baled at 2020.   |
| 2,020 | 2,028 | 8    | Brown limestone containing pyrite, anhydrite and sand. Some sulphur was deposited in the tube on heating.  |
| 2,028 | 2,035 | 7    | Brown limestone containing considerable sand, most of it below $\frac{1}{8}$ mm. and a little chert. On heating it gave off the odor of bitumen. An Endothyra was noted.   |
| 2,035 | 2,041 | 6    | Argillaceous dolomitic sandstone containing anhydrite and pyrite. The majority of the sand grains are smaller than $\frac{1}{8}$ mm. Some oil was distilled off on heating.  |
| 2,041 | 2,045 | 4    | Gray argillaceous dolomitic sandstone containing a black mineral, probably chert. Some oil was given off on heating. Water was encountered at 2045.  |
| 2,045 | 2,052 | 7    | Gray sandy dolomitic limestone containing a little anhydrite, pyrite and bitumen. A little oil was distilled off on heating.   |
| 2,052 | 2,138 | 86   | Gray sandy dolomitic limestone containing pyrite, chert and a little anhydrite. Gave off odor of bitumen on heating.   |
| 2,138 | 2,190 | 52   | Gray sandy limestone. Most of the sand is below $\frac{1}{8}$ mm. Some oil distilled off on heating.   |
| 2,190 | 2,197 | 7    | Blue-gray sandy dolomitic limestone. A little oil distilled off on heating.  |
| 2,197 | 2,204 | 7    | Gray limestone containing some sand, most of which is below $\frac{1}{8}$ mm. Some oil was distilled on heating.   |
| 2,204 | 2,230 | 26   | Gray sandy dolomitic limestone containing a little chert (?). Gave off a slight odor of bitumen on heating.  |
| 2,230 | 2,236 | 6    | Gray dolomitic limestone containing considerable sand, most of which is below $\frac{1}{8}$ mm., and black fragmentary mineral.  |

*Topography*

No topographic survey of this area has been made. As already observed, the main topographic feature of this entire region is the Guadalupe Mountains and their southward extension into the Delaware Mountains. The elevation of El Capitan Peak, Guadalupe Mountains, is taken at 8690 ft. (2648 m.) and this is probably the highest point in the State. From these mountains the eastward slope is about 100 ft. (30 m.) to the mile to the Pecos River, a distance of about 60 miles (96 km.). The Rustler Hills, somewhat nearer to the river than to the mountains, intervene between the river and the mountains. They extend in a general northeast-southwest direction for more than 40 miles and form the western boundary of the Maverick Springs district. Their maximum elevation is about 4000 ft. (1219 m.). At intervals they are cut through by east and northeast drainage lines, such as Cottonwood Draw, Rustler Draw, Horseshoe Draw, etc. Although the mean annual rainfall does not exceed 10 in. (254 mm.) a great deal of water passes down these Draws and there has been much erosion.

In the Maverick Springs district, the principal sulphur deposits now exposed are in a valley known as the Virginia Draw, lying on the east and northeast side of the Rustler Hills and extending from Little Rock Tank, in Section 30, Block 111, to and into Section 24, Block 113, a distance of about 7 miles in a northeast direction. This valley is bounded on the west and northwest by the Rustler Hills and on the east and southeast by a low range of hills characterized by rounded knolls of dolomite.

The drainage down this valley is, thus, from the Rustler Hills and the lower dolomite hills towards the northeast. In ordinary times there is no surface water in this valley at all, but after the infrequent rains there are numerous natural tanks and a few rock tinajas in which water accumulates and lasts for a while. The slope of the valley is about 30 ft. to the mile and the throw of the drainage is against the dolomite hills in the upper part and against the outcrops of massive gypsum and gypsite in the lower part. Northeast of the Dot property, the almost flat part of the valley widens out to as much as 1200 ft. (366 m.) but is much indented, the low escarpment on the west encroaching on the flat and then withdrawing, forming small embayments.

A good deal of underground water finds its way down this Draw and it is always accompanied by more or less hydrogen sulphide. This water comes to the surface in the lower part of the Draw, just before its junction with Salt Draw, which latter Draw takes the drainage from Rustler Draw, Horseshoe Draw, Horseshoe Springs, etc. On the west side of Virginia Draw there is a low gypsum mesa, with sporadic outcrops of



dolomite, and sulphur deposits have been found on this mesa, especially near the Dot property, at depths as low as 50 ft. (15 m.).

### *Climate*

The climate of this part of Texas is arid, the mean annual rainfall being between 8 and 10 in. (203 to 254 mm.). For long intervals there is no rain at all. During the winter months there may be a little snow or sleet, but it does not remain on the ground.

### *Water Supply of the District*

The shallow wells throughout the country yield an abundance of water, but is highly impregnated with mineral matter. In Table 1 are collected analyses of water from a number of different sources. In regard to sample No. 2, Richardson says:

"Delaware Springs are at the head of flowing water in Delaware creek. Here water issues along the north side of the creek at the contact of gravel and a sandstone member of the Delaware Mountain formation. The quality of the water from all of the above springs is good and the water is soft. \* \* \* But besides this good water several saline springs issue from the bed of Delaware creek in close proximity to the springs just mentioned having an altogether different and a deeper-seated source. The water in them bubbles up through joints in the Delaware Mountain formation that outcrops in the bed of the creek. This water contains considerable hydrogen sulphide, from which finely divided white sulphur is precipitated on contact with the air."

Dr. Booth mentions the presence of sodium sulphide in the water from a spring at the head of Delaware Creek and also in the water of the creek 30 miles below the springs and 10 miles from the mouth of the creek, where it empties into the Pecos River. He also found this ingredient in the water from Mustang Springs, Delaware Mountains. There can be no doubt of the existence of alkaline sulphides in the waters of the sulphur district and this fact has an important bearing on the origin of the sulphur deposits, as was observed by Blake more than 60 years ago.

Samples 6 and 7 taken in 1904 were from the same waters in Salt Draw as samples 8 and 9, but they were secured in February after a long period in which there was no rain. Samples 8 and 9 were taken in November after the fall rains and they show a lower concentration than the others. The absence of hydrogen sulphide from the running water is due to the oxidation of this compound. Free sulphur is now being deposited from these waters, especially at the Falls. Sample 8, from the pool at the head of water in Salt Draw, represents the underground drainage that comes into this Draw from Rustler Draw and Horseshoe Draw. Sample 9, of running water from Salt Draw, represents the water

TABLE 1.

| Sam-<br>ple No. | Ca                  | CaSO <sub>4</sub> | Mg     | MgSO <sub>4</sub> | MgCl <sub>2</sub>   | Na + K<br>(as Na) | NaCl      | Na <sub>2</sub> CO <sub>3</sub> | Na <sub>2</sub> SO <sub>4</sub> | Al <sub>2</sub> O <sub>3</sub> | As    | SiO <sub>2</sub> | SO <sub>3</sub>   | CO <sub>2</sub> | Cl      | S<br>(as H <sub>2</sub> S) | Total     |
|-----------------|---------------------|-------------------|--------|-------------------|---------------------|-------------------|-----------|---------------------------------|---------------------------------|--------------------------------|-------|------------------|-------------------|-----------------|---------|----------------------------|-----------|
| 1               | .....               | 218.26            | .....  | 11.25             | .....               | .....             | 23.10     | 4.83                            | .....                           | Tr. <sup>1</sup>               | ..... | 3.44             | .....             | .....           | .....   | .....                      | 260.88    |
| 2               | 6.64 <sup>a</sup>   | .....             | 5.89   | .....             | .....               | .....             | .....     | .....                           | .....                           | 0.32 <sup>1</sup>              | ..... | 0.28             | 2.46 <sup>a</sup> | .....           | .....   | .....                      | 184.53    |
| 3               | 9.351 <sup>10</sup> | 111.790           | .....  | .....             | 6.337 <sup>11</sup> | .....             | 20.364    | .....                           | .....                           | Tr. <sup>1</sup>               | ..... | 1.686            | .....             | .....           | .....   | .....                      | 183.637   |
| 4               | .....               | 105.53            | .....  | 11.26             | .....               | .....             | 6.82      | .....                           | 53.92                           | 22.73 <sup>a</sup>             | Tr.   | 12.82            | .....             | .....           | .....   | .....                      | 213.08    |
| 5               | 3.00 <sup>1</sup>   | 90.36             | .....  | 5.03              | .....               | .....             | .....     | .....                           | 61.55                           | 1,291.924                      | Tr.   | .....            | .....             | .....           | .....   | .....                      | 1,451.86  |
| 6               | .....               | 274.087           | .....  | 72.128            | 42.127              | .....             | 1,026.181 | 73.784                          | .....                           | 0.839 <sup>a</sup>             | ..... | 0.046            | .....             | .....           | .....   | .....                      | 1,489.192 |
| 7               | .....               | 234.202           | .....  | 124.737           | 32.578              | .....             | 1,309.011 | 126.493                         | .....                           | 1.445 <sup>a</sup>             | ..... | 2.705            | .....             | .....           | .....   | .....                      | 1,831.202 |
| 8               | 57.066              | .....             | 15.192 | .....             | .....               | 118.159           | .....     | .....                           | .....                           | .....                          | ..... | .....            | 154.336           | 6.185           | 209.477 | 1.967                      | 612.792   |
| 9               | 56.833              | .....             | 9.803  | .....             | .....               | 147.626           | .....     | .....                           | .....                           | .....                          | ..... | .....            | 149.551           | 6.837           | 242.153 | None                       | 1,640.553 |
| 10              | 80.756              | .....             | 22.408 | .....             | .....               | 498.192           | .....     | .....                           | .....                           | .....                          | ..... | .....            | 226.748           | 17.913          | 794.536 | .....                      | 1,225.198 |
| 11              | 73.696              | .....             | 17.681 | .....             | .....               | 349.517           | .....     | .....                           | .....                           | .....                          | ..... | .....            | 177.326           | 26.316          | 573.294 | 5.358                      | 1,225.198 |
| 12              | 66.986              | .....             | 35.243 | .....             | .....               | 498.309           | .....     | .....                           | .....                           | .....                          | ..... | .....            | 231.066           | 23.223          | 742.053 | 3.793                      | 1,600.473 |

## Description of Sample

Sample No. 1.—From the Cooksey ranch, Section 15, Block 114. Richardson.

Sample No. 2.—From a spring at the head of Delaware Creek. Richardson.

Sample No. 3.—From "Sinking Sheep," Section 9, Block 114, northwest of Cooksey ranch. Analysis by S. H. Worrell of sample by the writer, February, 1904. This water had a nauseating taste and carried considerable hydrogen sulphide. It had the local reputation of being poisonous, and at the time of the writer's visit, there were many bones of cattle, birds, etc., to be seen.

Sample No. 4.—From Section 11, Block 114, 25 ft. below surface. E. M. Skeets.

This water contained 79.08 grains of free sulphuric acid per gallon.

Sample No. 5.—Sample from the Dot property, in Virginia Draw, 10 ft. below the surface, carried still more free acid.

Sample No. 6.—From Salt Draw, north of house at Maverick Springs.

Analysis by S. H. Worrell from sample by the writer, February, 1904.

Sample No. 7.—Same, west of house. This sample contained much free hydrogen sulphide.

Sample No. 8.—From pool at head of water; Salt Draw. Analysis by W. T. Read from sample by the writer, Nov., 1916.

Sample No. 9.—Same from running water below the falls.

Sample No. 10.—From Virginia Draw, northwest part of Section 1. Block 111, 1 1/4 miles below the Dot property, at a depth of 23 ft. Analysis by W. T. Read, from sample by the writer. This sample smelled strongly of hydrogen sulphide, but was exposed to the air and the gas escaped, so that it does not appear in the analysis.

Sample No. 11.—From the Big Hole in Virginia Draw, near road from Maverick Springs to house of W. H. Hicks; 1 mile below the location of sample No. 10. Analysis by W. T. Read from sample by the writer.

Sample No. 12.—From seepage in a shallow pit at Maverick Springs. Section 18, Block 45. Analysis by W. T. Read from sample by the writer.

<sup>1</sup> Alumina and iron.

<sup>2</sup> Combined sulphuric acid.

<sup>3</sup> Al<sub>2</sub>(SO<sub>4</sub>)<sub>3</sub> + Fe<sub>2</sub>(SO<sub>4</sub>)<sub>3</sub>.

<sup>4</sup> Al<sub>2</sub>(SO<sub>4</sub>)<sub>3</sub>, 628.47; Fe<sub>2</sub>(SO<sub>4</sub>)<sub>3</sub>, 663.45.

<sup>5</sup> Phosphate of lime.

<sup>6</sup> This carries a trace of iron.

<sup>7</sup> Bicarbonate.

<sup>8</sup> MgO.

<sup>9</sup> CaO.

<sup>10</sup> CaS.

<sup>11</sup> Also 34.199 CaCl<sub>2</sub>.

after it has been mixed with the drainage from Virginia Draw. On the whole these waters are much alike. Sample 12, from Maverick Springs, is closely similar in composition to sample 10, obtained from a depth of 23 ft. (7 m.) in the Virginia Draw. The locality in this Draw is 2 miles (3.3 km.) southwest from Maverick Springs and is separated from Maverick Springs by the ridge that lies between Virginia Draw and Maverick Draw. As the water in the Virginia Draw is probably drainage from the sulphur deposits higher up the valley, so the water in the "seep" at Maverick Springs may be the drainage from the sulphur deposits that are west of the Springs.

There is abundance of ordinary "gyp" water in Maverick Springs at all times. It is the largest and most permanent water supply in the district. Above the Springs there is a large drainage area extending to and beyond Big Rock Tank, Section 26, Block 111. There are no known sulphur deposits in Maverick Draw and the water is as good as any in the country.

By constructing an earth dam below Maverick Springs, it would be possible to impound a large quantity of water, and it is thought that this will have to be done, if the district is developed. Of course, there is more or less water to be obtained from wells sunk in the Maverick Draw above the Springs.

The Michigan Sulphur & Oil Co. is now obtaining its water from a windmill in Rustler Draw, about 9000 ft. from the property, through a gravity pipe line. The supply is sufficient only for a small camp and a limited business.

It is uncertain whether artesian water suitable for use could be secured within reasonable drilling depths. For a part of the year, the time depending on the rains, "soft" water can be obtained from natural water holes, but this source of supply is too uncertain to be considered seriously.

### *Fuel*

The local fuel is juniper, or mountain (scrub) cedar. It is hauled in from points 5 to 10 miles distant (8 to 16 km.). It makes a quick and hot fire, but has no lasting qualities. The supply of this fuel is sufficient only for ordinary household purposes, cooking, heating, etc. It cannot be depended on for raising steam in a plant of any size.

Twenty miles (32 km.) southeast of the Maverick Springs district, in Section 16, Block 59, Reeves county, are the shallow oil wells on the Huling-White-Ross ranch. They have been known for some years, but have not been used as a source of oil except for lubricating windmills and so forth. The oil is an excellent natural lubricant and comes from a depth of about 230 ft. (70 m.).

Fuel for industrial purposes in this district will have to be brought

in over a branch line of railroad built from Orla, 15 miles from Maverick Springs. Best screened Texas lump coal can be laid down at Orla for \$5.15, the haul being 366 miles from Thurber.

There has been more or less talk about the natural gas in the district being sufficient for industrial purposes. There is little except hope on which to base such expectations. The gas that has been found there comes from shallow depths and no one competent to judge has so far expressed a favorable opinion on the possibilities of finding enough of it, in depth, to be of consequence. There is many a wish that is father to the thought.

### *Nature and Origin of the Deposits*

Skeats, in the bulletin previously referred to, describes at considerable length the topography and geology of the district. His observations on the origin of the sulphur deposits are particularly pertinent, and agree in great measure with those of William P. Blake, of the Pope Expedition, and with those of later geologists and engineers. In this connection he says:

"The origin of the sulphur is not easy to decide, for while it is generally thought to be of solfataric origin yet there is little evidence to support this opinion. All of the sulphur of the region occurs in and with gypsum and in connection with water containing sulphuretted hydrogen. This water has been traced by the writer from the upper end of Crow Flat, a valley on the east side of the Guadalupe Mountains.

"If decomposition of the sulphuretted hydrogen produced the sulphur, the deposits should occur over a wider area than is now thought to be the case. Sulphur may be produced by the alteration of gypsum, but in this case we would have to consider the nature of the reducing agent and explain why its action appears to be confined to a line near which passes an underground stream of sulphur water. From a careful study of the country and the processes now at work there, the writer is of the opinion that the richer bluish ores have been formed from sulphur waters at a time when they were above ground, and probably through the agency of certain algae which are plentiful in the sulphur springs today. The organic matter associated with these ores gives strength to this view, and also the fact that it is quite common to find the black gravel composed of small rounded pebbles in alternation with the ore. The ores in which the sulphur occurs as crystals are probably formed by the decomposition of sulphuretted hydrogen given off from the highly charged water when it enters a porous or broken stratum.

"In some places the ore may be a replacement product, the original gypsum having been removed. It is possible, also, that what is now gypsum was once carbonate of lime. Instances might be given of the occurrence of gypsum hills capped with limestone, or dolomite, which is in close association with underlying beds of gypsum. In many places it is difficult to say just where the limestone ends and the gypsum begins.

"These are points of great practical importance, for if we knew the mode of formation of the sulphur we would know where to look for it, and could form some idea of how much to expect. Only these deposits which have been uncovered by erosion have as yet been found, and it is likely that systematic prospecting would bring to light other deposits in the region."

In 1891, O. C. D. Ross presented a paper before the British Association for the Advancement of Science on the origin of sulphur, petroleum, etc.<sup>10</sup> Mr. Ross gave a series of hypothetical chemical reactions to show that carbonate of lime, sulphuretted hydrogen and peroxide of hydrogen could yield gypsum and hydrocarbons analogous to ethylene ( $C_2H_4$ ). Mr. Ross's theory is ingenious but does not rest upon any known facts.

The possible influence of sulphur-secreting algæ on the origin of sulphur is also interesting. The Beggiatoa, or sulphur-bacteria, contain grains of sulphur and these may have arisen through the oxidation of sulphuretted hydrogen in a manner comparable with the power of Crenothrix and Leptothrix ochracea (iron bacteria) to oxidize ferrous compounds to ferric compounds. The disappearance of the sulphur grains in the Beggiatoa has been thought to be due to their oxidation to sulphuric acid.

With reference to the influence of certain algæ, mentioned by Skeats as occurring in the sulphur waters today, Dr. Emile Böse, of the Bureau of Economic Geology and Technology, University of Texas, suggested that the small holes in the magnesian limestone of the Rustler formation may be the remains of groups of algæ. These markings, mentioned by Richardson, on page 44 of his report, occur so plentifully as to suggest a vigorous and continuous growth. They vary in diameter from a fraction of an inch to more than an inch and are of circular cross-section. For the most part they are now well crystallized calcite. They occur in great abundance on a small dolomite hill near the center of Section 1, Block 111, as also in slabs scattered on the surface of the mesa between Virginia Draw and Lindsey's house. Richardson speaks of them as occurring in a section of 50 ft. (15 m.) in Horsehoe Draw, associated with a calcareous buff sandstone and Castile gypsum. Bischof<sup>11</sup> gives many instances of the transformation of limestone into gypsum through the action of sulphuretted hydrogen (hydrogen sulphide). Thus, Covelli found beautiful crystals of gypsum and sulphur in the crater of Vesuvius. In examining the fumaroles of Tuscany, Dumas found no free acid and he ascribed the alteration of the limestone there into gypsum to the action of sulphuretted hydrogen and connects this with a similar phenomenon at the sulphur baths, near Aix, where the walls of limestone gradually become covered with crystals of gypsum. Hoffman found extensive deposits of gypsum in the Lipari Islands, formed through the action of sulphuretted hydrogen on limestone, and gives an instance of the occurrence of gypsum in tuff beds underlying lava. Not only are limestones thus decomposed, but even silicates. There is abundant

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<sup>10</sup>An abstract of the original paper appeared in *Chemical News* (1891), **64**, 191, and Mr. Boverton Redwood commented on it in the same volume, page 215.

<sup>11</sup>*Chemical and Physical Geology*.

evidence of the formation of gypsum and sulphur through the action of sulphuretted hydrogen on limestone. Bischof says:

"The simultaneous formation of gypsum and separation of sulphur may be effected artificially by bringing sulphuretted hydrogen in contact with moist chalk. When this gas is heated up to 212° F., sulphur is deposited in the conducting tube, where it comes in contact with the air; it may, therefore, be readily understood that moderately heated sulphuretted hydrogen, continually streaming into a bed of limestone, may give rise to the formation of gypsum and the deposition of sulphur, when atmospheric air is not excluded. While, on the one hand, sulphur is separated from gypsum which is saturated with bituminous substances by a simple decomposition, sulphuretted hydrogen streaming through beds of limestone gives rise to the simultaneous production of gypsum and sulphur; the very association of these two substances admits of no other explanation."

It would appear that the sulphur deposits in Culberson county may have arisen in one or more of the following ways:

1. Through the action of hydrogen sulphide on limestone or dolomite.
2. Through the decomposition of waters carrying alkaline sulphides.
3. Through the decomposition of gypsum impregnated with bituminous substances.
4. Through the precipitation of sulphur from sulphur-bearing gases, especially sulphuretted hydrogen.

The conditions under which these factors have operated are observable in the district today, but they may have been more active and intensive than they are now.

It is not probable that volcanic or seismic agencies are responsible for the deposition of this sulphur, although they may have been influential in causing or accelerating dislocations or alterations of the strata and may thus have provided for the circulation of sulphur-bearing waters and gases. It is possible that the alteration of gypsum impregnated with bituminous substances contributed to the formation of some of the deposits, for there are many localities where bituminous gypsum is observable in close association with sulphur. This is particularly the case at the Grant mines, Sections 17 and 18, Block 61, and on Section 1, Block 46, where there are heavy beds of a wavy gypsum carrying a dark brown bituminous substance.

### *Utilization of the Deposits*

So many factors enter into the utilization of these sulphur deposits that to discuss them in detail would protract this paper to an undue length. The principal ones are:

The nature and thickness of the overburden.

The nature and thickness of the several bands of sulphur-bearing material that are within workable distances from the surface.

The methods of extraction.

Transportation of the product to available markets.

Sale of the product, inclusive of the capital required for carrying stocks.

The development work that has been done is not sufficient in character or extent to allow one to answer any one of these questions as fully as could be desired. While the writer believes that some of the deposits are well within commercial possibilities, this belief is based more upon his knowledge of such deposits in general than upon the results of detailed prospecting and development on any particular area. In the district as a whole the work already performed has not been conducted systematically. The statements of millions of tons of workable material have not been confirmed by regular sampling and analysis. Entirely too much has been assumed, both with respect to the thickness of the several strata carrying sulphur and the composition of these strata. The writer has before him now a statement that on such and such a holding there are more than one million tons of 99.85 per cent. pure sulphur and that a considerable acreage represents more than two million tons of pure sulphur. Statements of a like nature with respect to other properties have a similar basis of hope. It is idle to consider such reports as forming a solid foundation on which actual development work may be undertaken. With the best intentions, no engineer or geologist who is careful of his reputation can afford to certify to anything more than has been determined with reasonable accuracy. In deposits of this character, subject to considerable variations in horizontal extent, depth and content of recoverable values, there is nearly always a margin of risk which must be taken into consideration. It is the purpose of systematic prospecting to reduce this risk to a minimum, but it can never be wholly removed, and the degree of the removal is not always determinable.

The advantages that this sulphur district may have are: (1) it is 700 miles nearer the western points of actual and possible consumption than the deposits in Texas at the mouth of the Brazos River; and (2) the initial investment would be comparatively small with respect both to prospecting and development.

No reliable data can now be given on the cost of production. The operations have been conducted on so small a scale and so intermittently that the cost accounts that have been kept are not reliable. The same observation applies, on the whole, to the quality of the material obtainable. Instead of a scant 20 or 30 analyses there should be hundreds, following careful sampling. Whether the available material carries 25 or 30 or 35 per cent. of sulphur is not known. Picked specimens of 50, 60 and even 90 per cent. sulphur cut no figure, except a delusive one. They do not mean anything to the engineer engaged in a critical examination of actual conditions. Such hand specimens are nice to have

around and to send to museums, etc., but when it comes to real business they are very good things to let alone.

In summing up this paper I wish to say that some of the sulphur properties at and near Maverick Springs and towards the upper part of Virginia Draw seem to be worthy of systematic prospecting. There is a considerable amount of sulphur in this district within 50 ft. (15 m.) of the surface and it can be mined cheaply. I am disposed to take the average content of sulphur in the crude material as not exceeding 30 per cent., although there are strata of higher sulphur content. That is to say, the average content of native sulphur in the material that can be mined and sent to the extractor will be around 30 per cent. There will doubtless be strata of a higher content and strata of a lower content.

It is not known today how much of the area involved will yield material carrying 30 per cent. of sulphur, nor what is the average thickness of the overburden that will have to be removed.

I take it that practically all of the work in mining the sulphur will be by the open-pit and bench system, the overburden being removed and sent back to allow of the full lift of the shovel. This system could be used to a depth of 50 ft.

Underground operations would involve a considerable expense for timbering and ventilation, and are not now to be seriously considered. Looking at the matter as a whole, I do not think that plans for actual development can now be made owing to the lack of systematic prospecting, and it is too early in the game to plan for extraction plants. Such a plant is the last thing to be taken up. Its location, design and equipment depend on factors not yet clearly understood. It requires a good deal of money to go into the sulphur business and it requires a good deal to stay in it, to be ready to take contracts and to fill them promptly. At the same time, and knowing the situation in Culberson County thoroughly, I am of the opinion that some of the sulphur deposits there are worthy of attention as a continuous source of sulphur of good quality.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

## A Uniform Sizing Diagram from Different Screen Standards

BY JOHN RANDALL,\* BOULDER, COLO.

(St. Louis Meeting, October, 1917)

It is a fair assumption that the main purpose of any diagram is to present facts to the eye in more convenient form than they could be tabulated in figures, and this implies that a screen diagram should set forth in a convenient and familiar form certain facts as to the character of the material screened regardless of the screen standards used, and it seems particularly desirable on account of the well-known differences in the size of openings of different laboratory screens having the same nominal mesh.

In making screen-sizing tests it has been customary to first consider the largest size, going progressively to the smallest, and this seems to have led to the practice of placing the smallest size at the right-hand side of the diagram and measuring toward the left. This is not altogether convenient and might be changed to advantage, but for illustration I have preserved that feature and present the method just as I have used it for over 2 years. Direct plotting in which the intervals on the diagram are proportional to the sizes of the openings, or to the arithmetical difference between them, seems at first sight more natural and is extremely useful for certain purposes, but it makes the diagram very crowded in the smaller screen sizes while the larger ones occupy unnecessary space, making it less convenient for most purposes, and for that reason the method here discussed is the well-known one based on the ratio between openings, and is plotted in logarithms instead of the direct values of the screen opening. Metric coordinate paper is convenient, although not necessary; it is recommended, as the use of the metric scale gives a diagram of convenient size, but any scale of equal parts may be used.

In order to make any screen diagram translatable in terms of any standard screen size, it is necessary to have noted on the size scale of the diagram all points corresponding to the openings of the various screens in question, instead of a single series of points as with a single screen. In drawing the diagram assume 0.001 in. (corresponding approximately to 0.025 mm.) to be the smallest size to be considered and fix this as the zero point at the lower right-hand corner of the diagram. This value is selected because the values are to be plotted in logarithms, and the logarithm of 1 is zero. Then lay off to the left for the position of each screen

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\* Metallurgical Engineer.

opening a distance on the scale equal to the logarithm of the opening. If the screen openings are given in millimeters instead of inches, a corresponding change must be made in the horizontal scale of the diagram. The zero point of the diagram corresponds to 0.025 mm. and since the mantissa of the logarithm of 0.025 is approximately 4, the logarithmic values for the millimeter scale can be made directly comparable with the inch scale by subtracting 4 from the first figure of the mantissa of the millimeter logarithm. After the subtraction of 4 the screen openings can be located on the scale as before and their position will coincide with that of openings of the same size given in inches. This being the case, it is immaterial what unit of measurement is used in designating the screen openings, since it is very easy to transfer directly from one scale to the other. This transfer is of course not absolutely accurate but it is sufficiently accurate for the purposes of the comparison.

In Table 1 is given the screen openings and logarithm values for four different sets of laboratory screens, the logarithms being rounded off to not more than two decimal places for convenience in plotting, and the characteristics being omitted. The rounding off of the logarithms makes the ratio exact in the case of the screens having a common ratio of increase of the size of the opening, a condition not otherwise capable of exact expression decimally, and introduces no appreciable error. The different screens here tabulated in part are: (1) The Tyler screens, based on the opening of a 200-mesh screen standardized by the U. S. Bureau of Standards, only each alternate size being here listed, the lineal dimensions of the openings having an ascending geometric ratio of 2; ratio with all the sizes in would be  $\sqrt{2}$ ; (2) the I. M. M. screens, having the sanction of the Institution of Mining and Metallurgy, London, and almost universally used throughout the British Empire. The screens here shown approximate for practical purposes a ratio of 1.5874, or  $\sqrt[3]{4}$  for the ratio of increase. Stadler (Royal School of Mines, So. Kensington, London) has proposed that these screens be manufactured in the following sizes in order to make the ratio more nearly exact, but as now made the approximation to a common ratio is close:

*Screen Openings Proposed by Stadler*

| Nominal Mesh | Diameter of Opening in Inches |
|--------------|-------------------------------|
| 12           | 0.03937                       |
| 20           | 0.02480                       |
| 30           | 0.01562                       |
| 50           | 0.00984                       |
| 80           | 0.00620                       |
| 120          | 0.00391                       |
| 200          | 0.00246                       |

When the I. M. M. screen standard was first adopted, there seems to have been no attempt to secure a common ratio between openings, as

the set originally contained a number of odd intermediate sizes, but the importance of a common ratio is now well recognized in England and is secured with substantial accuracy by using only a part of the original set.

3. The third set of screens here listed is by a well-known American maker. They have no common ratio between openings, but are considerably used in this country.

4. The fourth set is one lately recommended by the U. S. Bureau of Standards after much study and conference with makers and users of screens. The 0.062-mm. screen is almost identical with the I. M. M. 200-mesh screen and for practical purposes these screens coincide with other screens at a number of points.

To locate on the diagram (Fig. 1) the screens enumerated, we will refer to the logarithms in Table 1. The Tyler 200-mesh, since  $\log .29 =$

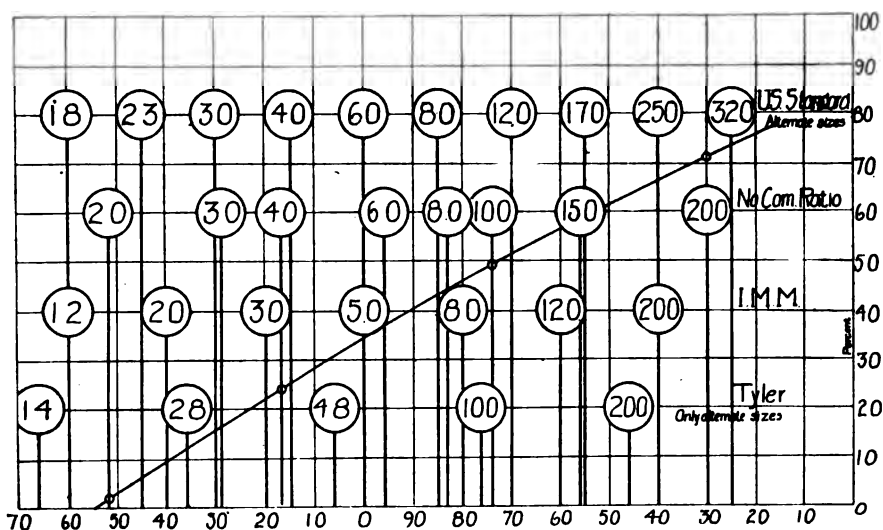


FIG. 1.

46, will be 46 scale units from the right index, and each succeeding screen will be 30 units farther to the left. In like manner, the 200-mesh I. M. M. screen will be at 40 and the 200-mesh screen of the set having no common ratio will be at 30 from the right. Dropping 4 from the first figure of the mantissa, we find that the logarithm, of the dimensions given in millimeters exactly check with the points already located, and we may proceed to locate the points for the fourth set. It will be noticed that the errors introduced to make the logarithms in exact arithmetical progression are slight. For instance, in the logarithm column for the new U. S. standard there is an error amounting to 0.412 scale unit at the last screen, which theoretically has an opening of 0.0625 mm., and this error

is thrown to the other side of the diagram by using the even figure 40 for placing the plotting scale instead of the more nearly correct 40.4834, the logarithm of the exact millimeter equivalent of 0.001 in. This, however, is considerably within the limit of tolerance allowed. It will be noticed that the indices of the logarithms change at each 100 mm. of the scale along the *A*-axis and that same distance along the *B*-axis of the diagram seems a most convenient range for the percentages, 0 to 100, of the material screened. This proportion between the dimensions along the two axes of the diagram is now much used and leaves nothing to be desired.

It is hardly necessary to add that the most common method of plotting is to locate the curve at the points along the *B*-axis at which all the material remains on a given screen, thus making the diagram cumulative. The curve drawn for illustration is from a stamp-battery product and only four screens were used.

It is probably too much to hope that everybody will eventually use the same screen standard, but the adoption of a plan giving the same curve for similar materials regardless of screen standards would be a welcome improvement. It is hoped that these suggestions will at least be helpful to those who might desire to change to a better screen standard and at the same time have a convenient means of comparing former work.

TABLE 1.—*Screen Openings*  
Tyler Screens (W. S. Tyler Co., Cleveland)

| Nominal Mesh | Opening in Inches | Logarithm of Opening Rounded Off | Opening in Millimeters | Logarithm (Metric) |
|--------------|-------------------|----------------------------------|------------------------|--------------------|
| 8            | 0.0928            | 0.96                             | 2.362                  | 0.360              |
| 14           | 0.0464            | 0.66                             | 1.168                  | 0.060              |
| 28           | 0.0232            | 0.36                             | 0.589                  | 0.760              |
| 48           | 0.0116            | 0.06                             | 0.295                  | 0.046              |
| 100          | 0.0058            | 0.76                             | 0.147                  | 0.160              |
| 200          | 0.0029            | 0.46                             | 0.074                  | 0.860              |

I. M. M. Screens (Inst. Mining & Metallurgy, London)

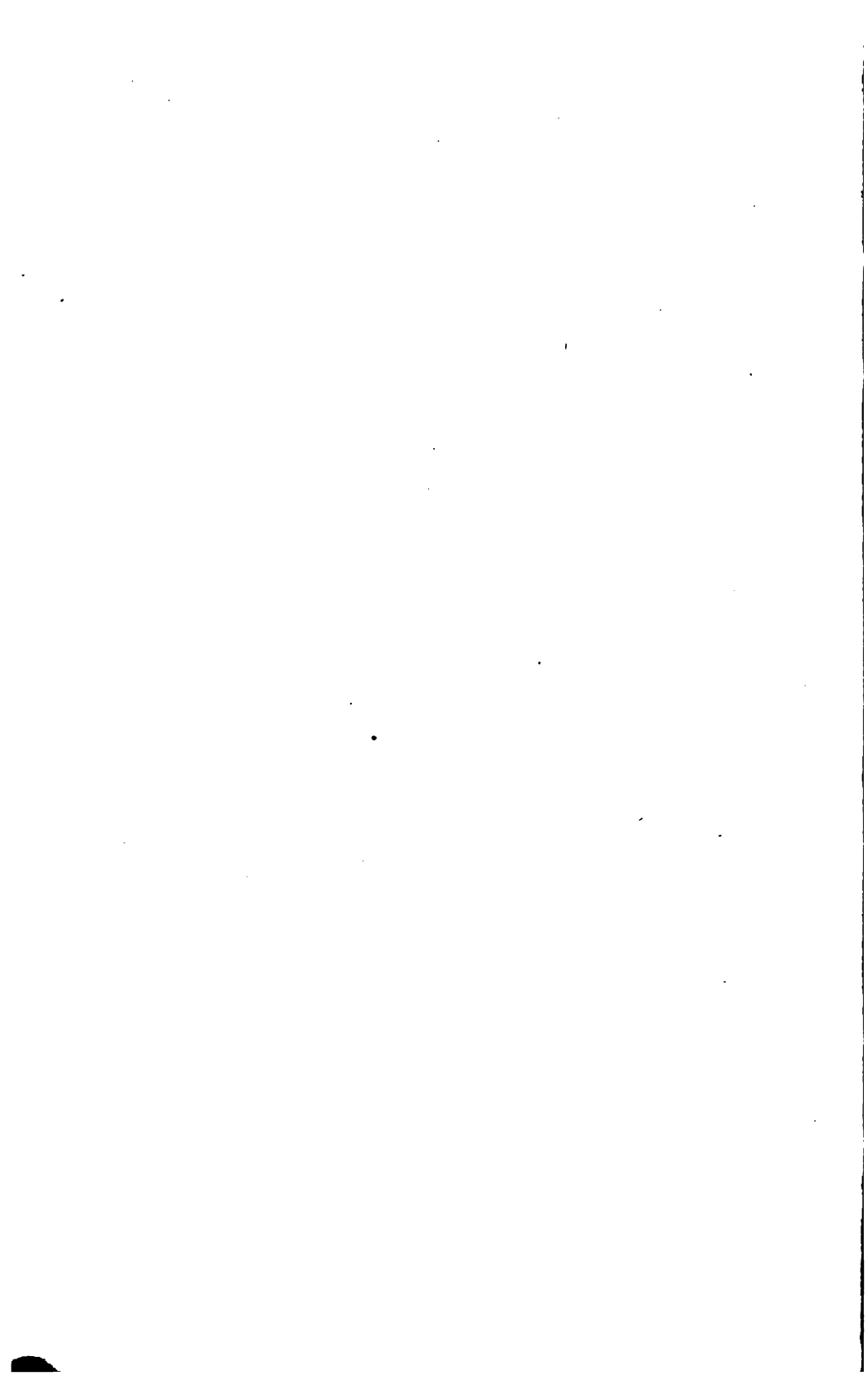
|     |        |      |       |      |
|-----|--------|------|-------|------|
| 12  | 0.0416 | 0.60 | 1.056 | 0.00 |
| 20  | 0.0250 | 0.40 | 0.635 | 0.80 |
| 30  | 0.0166 | 0.20 | 0.421 | 0.60 |
| 50  | 0.0100 | 0.00 | 0.254 | 0.40 |
| 80  | 0.0062 | 0.80 | 0.157 | 0.20 |
| 120 | 0.0042 | 0.60 | 0.107 | 0.00 |
| 200 | 0.0025 | 0.40 | 0.063 | 0.80 |

TABLE 1.—*Screen Openings.—(Continued)*  
Screens Having no Common Ratio

| Nominal Mesh | Opening in Inches | Logarithm of Opening Rounded Off | Opening in Millimeters | Logarithm (Metric) |
|--------------|-------------------|----------------------------------|------------------------|--------------------|
| 10           | 0.0799            | 0.90                             |                        |                    |
| 20           | 0.0335            | 0.52                             |                        |                    |
| 30           | 0.0195            | 0.29                             |                        |                    |
| 40           | 0.0147            | 0.17                             |                        |                    |
| 60           | 0.0091            | 0.96                             |                        |                    |
| 80           | 0.0067            | 0.83                             |                        |                    |
| 100          | 0.0055            | 0.74                             |                        |                    |
| 150          | 0.0036            | 0.56                             |                        |                    |
| 200          | 0.0020            | 0.30                             |                        |                    |

Screens Recommended by U. S. Bureau of Standards

| Approximate Mesh |  |  |       |       |
|------------------|--|--|-------|-------|
| 18               |  |  | 1.000 | 0.000 |
| 20               |  |  | 0.850 | 0.925 |
| 23               |  |  | 0.710 | 0.850 |
| 25               |  |  | 0.590 | 0.775 |
| 30               |  |  | 0.500 | 0.700 |
| 35               |  |  | 0.420 | 0.625 |
| 40               |  |  | 0.360 | 0.550 |
| 50               |  |  | 0.290 | 0.475 |
| 60               |  |  | 0.250 | 0.400 |
| 70               |  |  | 0.210 | 0.325 |
| 80               |  |  | 0.170 | 0.250 |
| 100              |  |  | 0.140 | 0.175 |
| 120              |  |  | 0.125 | 0.100 |
| 150              |  |  | 0.105 | 0.025 |
| 170              |  |  | 0.088 | 0.950 |
| 200              |  |  | 0.074 | 0.875 |
| 250              |  |  | 0.062 | 0.800 |



### The Function of Alumina in Slags

Discussion of the paper of CARL HENRICH, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 2081 to 2086.

J. E. JOHNSON, JR., New York, N. Y. (communication to the Secretary\*).—The discussion of Mr. Henrich's paper took place at a meeting from which I had the misfortune to be absent, and has just come to my attention. I do not wish to let the subject pass without contributing to it a little information which has come into my possession.

In 1912, I published a brief note in the *Transactions* on The Effect of Alumina in Blast-Furnace Slags, giving in tabular form three typical slags used in iron blast-furnace practice, one of them the high alumina slag of which Professor Richards speaks. That table is reproduced herewith.

TABLE 1.—*Composition of Slags*

|                        | 1  | 2                                  | 3                                     | 4                                      | 5   | 6   | 7                                |
|------------------------|--|------------------------------------|---------------------------------------|--|---|---|----------------------------------|
|                        | Al <sub>2</sub> O <sub>3</sub> ,<br>Per<br>Cent. | SiO <sub>2</sub> ,<br>Per<br>Cent. | CaO by<br>Difference,<br>Per<br>Cent. | Neutral<br>Substances,<br>Per<br>Cent. | Ratio<br>CaO<br>Al <sub>2</sub> O <sub>3</sub> + SiO <sub>2</sub> | Ratio<br>CaO + Al <sub>2</sub> O <sub>3</sub><br>SiO <sub>2</sub> | Ratio<br>CaO<br>SiO <sub>2</sub> |
| Virginia practice..... | 6.5  | 36.0                               | 54.0                                  | 3.5                                    | 1.27  | 1.68  | 1.50                             |
| Lake-ore practice..... | 13.5   | 33.0                               | 50.0                                  | 3.5                                    | 1.08  | 1.92  | 1.51                             |
| Special experiment...  | 36.0   | 24.7                               | 36.8                                  | 3.5                                    | 0.59  | 2.90  | 1.44                             |

It supplies what seems to me convincing evidence that, for the purposes of the iron blast furnace, alumina can be regarded only as a neutral substance, dissolving in the slag and materially altering its viscosity, but not affecting its chemical activity.

In subsequent experience with charcoal furnaces, I found the same rule to hold; that the chemical effect of alumina could be disregarded, providing the ratio of the lime and magnesia to the silica were kept constant. This is exactly the experience which Messrs. Dwight and Mathewson had in copper- and lead-smelting furnaces.

I have never seen any evidence or argument adequate to support the contention of Professor Richards—that the alumina acted as a base in the charcoal iron furnace and as an acid in the coke furnace, and that the function of alumina in iron-furnace slags is inherently different from that in copper- and lead-furnace slags—and I am compelled to differ absolutely with his conclusions in that respect.

I am not familiar with lead and copper furnaces, but certainly the

\* Received June 27, 1917.

experiences of Messrs. Dwight and Mathewson, and to a great extent the data of Mr. Henrich himself, might be taken as the description of my own experience with iron furnaces.

The rôle of alumina appears to be a purely physical one, but is highly important in that respect since it alters essentially the viscosity of the slag. Moreover, its effect in this respect is not constant in iron-furnace slags. The viscosity of the slag rises as the alumina rises, from 5 or 6 per cent. to about 20 per cent., and the viscosity falls from 23 or 24 per cent. to 36 per cent. (which is as far as my information goes). The high-alumina slags are distinctly more fusible than those in the range from 20 to 25 per cent., and about as fluid as those around 12 per cent.  $\text{Al}_2\text{O}_3$ , which are in common use.

The clever device of Mr. Feild, described in his paper at the same meeting, furnishes us the means of really measuring accurately viscosity at varying temperatures, and by the use of his method, I hope to see in a few years the solution of the slag problems which have bedevilled so many metallurgists.



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|------------------------|-------|-------|-------|-------|-------|-------|--------|
| Speed,<br>Ft. per Min. | H.P.  | H.P.  | H.P.  | H.P.  | H.P.  | H.P.  | H.P.   |
| 400                    | 1     | 1½    | 2     | 2½    | 3     | 4     | 5      |
| 600                    | 1½    | 2¼    | 3     | 3¾    | 4½    | 6     | 7      |
| 800                    | 2     | 3     | 4     | 5     | 6     | 8     | 9      |
| 1000                   | 3     | 3½    | 5     | 6     | 7½    | 10    | 12     |
| 1200                   | 3¼    | 4½    | 6     | 7     | 9     | 12    | 15     |
| 1500                   | 3¾    | 5¾    | 7½    | 9     | 11½   | 14½   | 19     |
| 1800                   | 4½    | 6¾    | 9     | 10½   | 12    | 16    | 20     |
| 2000                   | 5     | 7½    | 10    | 14    | 15    | 17½   | 22     |
| 2400                   | 6     | 9     | 13    | 15    | 18    | 19½   | 26     |
| 2800                   | 6½    | 10    | 13¾   | 16    | 18½   | 22    | 28     |
| 3000                   | 7     | 11    | 14    | 16½   | 19    | 24    | 30     |
| 3500                   | 8     | 12    | 14½   | 17    | 19½   | 26    | 32     |
| 4000                   | 8¼    | 13    | 15    | 18    | 24    | 30    | 40     |
| 4500                   | 8½    | 13½   | 20    | 22    | 26    | 35    | 45     |

*Approximate Horsepower Transmitted by Leather Belts (Double)*

| Width of Belt    | 4-in. | 6-in. | 8-in. | 10-in. | 12-in. | 14-in. | 16-in. | 18-in. | 20-in. | 22-in. | 24-in. |
|------------------|-------|-------|-------|--------|--------|--------|--------|--------|--------|--------|--------|
| Speed,<br>F.P.M. | H.P.  | H.P.  | H.P.  | H.P.   | H.P.   | H.P.   | H.P.   | H.P.   | H.P.   | H.P.   | H.P.   |
| 400              | 2¾    | 4½    | 6     |        |        |        |        |        |        |        |        |
| 600              | 4½    | 6½    | 7     |        |        |        |        |        |        |        |        |
| 800              | 5     | 8½    | 11    |        |        |        |        |        |        |        |        |
| 1000             | 7½    | 11    | 14    |        |        |        |        |        |        |        |        |
| 1200             | 8     | 13    | 17    |        |        |        |        |        |        |        |        |
| 1500             | 10¾   | 16½   | 20    | 25     | 36     | 40     | 50     | 65     | 70     | 77     | 84     |
| 1800             | 13    | 18    | 24    | 30     | 40     | 50     | 60     | 75     | 85     | 90     | 120    |
| 2000             | 14½   | 20    | 28    | 35     | 45     | 60     | 65     | 80     | 90     | 120    | 130    |
| 2400             | 17¼   | 25    | 34    | 40     | 50     | 70     | 80     | 90     | 110    | 130    | 144    |
| 2800             | 19    | 30    | 35    | 50     | 65     | 80     | 90     | 95     | 130    | 140    | 165    |
| 3000             | 19½   | 32    | 36    | 55     | 70     | 90     | 95     | 125    | 150    | 170    | 180    |
| 3500             | 20    | 34    | 37    | 60     | 80     | 100    | 125    | 155    | 160    | 180    | 200    |
| 4000             | ....  | 36    | 56    | 65     | 85     | 125    | 150    | 160    | 170    | 200    | 240    |
| 4500             | ....  | 38    | 65    | 70     | 100    | 150    | 160    | 165    | 175    | 230    | 250    |



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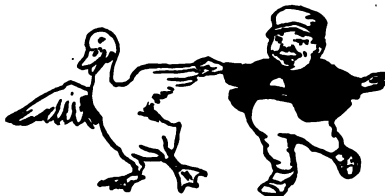
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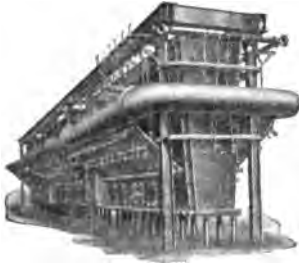
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